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A TEXT-BOOK OF ORE DRESSING

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A TEXT-BOOK

OF

ORE DRESSING

BY

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TO ALMIGHTY GOD;
THE FATHER OF OUR LORD JESUS CHRIST.

PREFACE

This book was written primarily to go with the Course on Mineral Dressing at the Royal School of Mines. Hitherto, the well-known works of R. H. Richards, H. Louis, and E. S. Wiard have been recommended for reference purposes, but with Flotation established as a prime process of mineral dressing these works no longer appear quite adequate. In these circumstances I have felt it a duty to make the necessary effort, the outcome being a book which, I venture to hope, may be of service to my own students and to others.

My competence in the subject lies to some extent in my own work and observation, but much more in the study of the published work and experience of others. Teachers and students to-day have at their disposal valuable Transactions of Mining and Metallurgical Institutions, and admirable technical Periodicals; upon such I have largely drawn, making endeavour always to give specific acknowledgement. Beyond this, to cover all the indebtedness for which no specific reference is possible, and such references as have unwittingly been overlooked, I gratefully make general acknowledgement.

I am likewise greatly indebted to my colleagues of the Imperial College, and particularly to Professor L. H. Cooke, and Mr. B. W. Holman, of the Mining Department, for advice and assistance, always insisting, however, that my good colleagues are not involved in any such crudities of statement or treatment as may be deemed to exist. To Mr. H. G. Dines, also, I desire to express my thanks for valuable help with the illustrations.

Though Ore Dressing does not cover the whole subject of Mineral

Dressing, its principles are the same as those applied in the dressing of non-metalliferous minerals, including coal; in addition, while Ore Dressing is relatively elaborate, non-metalliferous dressing is relatively simple. It is hoped eventually to include a description of coal-washing, to which and to non-metalliferous dressing there are in the present work but few references.

S. J. TRUSCOTT.

London, *January* 1923.

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CHAPTER I

INTRODUCTION ; PRINCIPLES

Nature and Scope.—"Ore" is metalliferous rock from which metal or metallic compound may be extracted commercially. Being such, it is the raw material of the metalliferous industry.

In the crude state as mined, such ore generally consists of: the metalliferous mineral, that is the ore-mineral, this being the valuable portion; the genetically associated but generally non-metalliferous mineral, the gangue-mineral; and a proportion of the containing rock, the country-rock. These two last are conveniently included under the term "gangue," while the ore-mineral may be described as "mineral." In this complex condition crude ore is generally unsuited for the extraction of metal or metallic compound by metallurgical operations; in addition, much of it is in large lumps, whereas metallurgical operations usually demand a smaller condition, sometimes even that of impalpable powder.

The operations which result in removing the gangue, as far as necessary or expedient, together with those which bring about the necessary reduction in size, constitute the art of ore-dressing. Such operations come between those of mining on the one side and those of metallurgy on the other, to prepare the ore delivered by the former for submission to the latter, or exceptionally for direct use in the industries.

Ore-dressing therefore embraces that series of preparatory operations to which the crude ore is submitted till any further work to extract the metalliferous content is best conducted metallurgically. The two arts have this relation: dressing operations effect a mechanical and preparatory concentration of the valuable constituents, whereas metallurgical operations effect a chemical and final concentration.

The place of dressing, therefore, is definite. It receives the crude ore from the mine and delivers it suitably cleaned for metallurgical treatment, or for use. On the side of the mine the line between mining and dressing is distinct: dressing takes the ore upon arrival at the shaft-top. On the side of metallurgy, though dressing is essentially mechanical and metallurgy essentially chemical, there is some overlapping of the two arts. Ore-dressing is, for instance, taken to include such calcining or roasting as may be necessary to induce magnetism preparatory to magnetic separation; such as may be necessary to break up a sulphide or an arsenide preparatory to subsequent water separation; and such as may be employed to sulphatize the surface of a sulphide in preparation for differential flotation. Roasting, of course, effects a chemical change and is in consequence primarily a metallurgical operation, but where it is preparatory to a directly-following mechanical or physical operation it presents itself for consideration with ore-dressing, as under similar circumstances do drying, and other heat treatments.

The preparatory character of these operations is not well disclosed by the name under which they go. In the term "*Dressing*" the suggestion of preparation is not nearly so implicit as that in the German "Aufbereitung" or the French "préparation mécanique," this latter suggesting, in addition, the mechanical nature of the operations. Frequently this preparation consists almost entirely of breaking, crushing, and grinding the crude ore to the fine condition necessary for the intimate working of extracting agents, while nearly always such comminuting operations play a considerable part. Ore-dressing operations are accordingly often described under the term "*Milling*"—a mill being primarily a building with machinery for grinding—and the whole range of the ore beneficiation is often given as mining, milling, and smelting or cyaniding as the case might be. Milling is undoubtedly one of the two primary operations of dressing, and sometimes, as in the preparation of ore for hydro-metallurgical processes, cyanidation for instance, it is the only operation. But in the preparation for pyro-metallurgical treatment, smelting for instance, the removal of gangue and consequent concentration of the mineral might necessitate more machinery than any necessary comminution; the works would then be spoken of as a concentrator rather than a mill and dressing reasonably described as "*Concentrating.*"

Necessity for Dressing.—The necessity for dressing lies in the physical and mineralogical differences between crude ore and ore suitable for metallurgical treatment or for direct use.

The most obvious difference lies in the size of individual pieces. The broken ore sent from underground to the surface contains pieces of all sizes, from huge lumps to the finest particles; yet for successful smelting no large lumps can be permitted, a common limit being about 4 in., and for hydro-metallurgical operations fineness is always necessary, sometimes even complete pulverization.

Not less important is the difference due to the low mineral-content of crude ore. Copper ore, for instance that from the large disseminated deposits, may contain as little as 1—1·5 per cent of copper, yet in copper smelting, by which the great bulk of metallic copper is recovered, a content rarely as low as 2·5 per cent and generally above 5 per cent is demanded, or otherwise there is too great a loss in the slag. The same may be said of lead; ore from the disseminated-lead deposits may contain as little as 3—4 per cent of lead, whereas lead smelting is rarely undertaken upon material containing as little as 10 per cent, a smaller percentage not allowing the proper collection of the silver generally present. With zinc it is the same; a large portion of the world's production of zinc is derived from ore containing but 4—5 per cent of zinc, whereas successful retorting requires the content to be not less than 30 per cent, or incipient fusion would take place and the distillation of the zinc would suffer. With tin the case is extreme; the broken ore does not usually contain more than 1 per cent of tin, yet smelting operations for the production of the metal are rarely conducted upon ore containing less than 55 per cent, because tin so readily slags with any silica present, and because other metals present in the impurities would in part also be reduced and contaminate the tin.

The case of iron is interesting. The dressing of iron ore was till within the last few years rarely practised, but to-day, with the demand for iron increasing at a rate the purer ores can no longer meet, attention is being turned to the poorer deposits and dressing is more common; ore containing 25 per cent of iron is now being mined, whereas ore with 50 per cent is as low as generally can be smelted. With the basic or Thomas process silica is harmful and must be removed, and with the acid or Bessemer process lime and phosphorus.

With the precious metals, gold and silver, when these are associated with pyritic sulphides, the concentration of such sulphides into a much smaller bulk would sometimes permit their submission to a metallurgical treatment impossible of application to the great bulk of the original crude ore, by reason of expense. Preparatory dressing would in such cases be a necessity.

Into this question of difference in valuable content, ordinary commercial economy enters. As would be expected, dressing once applied to meet technical necessity is not stopped when the barely-sufficient concentration has been reached, but is carried to its economic limit. With copper, for instance, though ore containing as little as 2·5 per cent may exceptionally be successfully smelted, a better recovery is obtained when the content is higher; some copper is always lost in the slag, and the amount so lost increases with the proportion of gangue, that is to say, with the lower value of the material smelted. Moreover, from the point of view of economy, smelting is not generally possible at the mines themselves; it demands cheap fuel, cheap fluxes, good smelting mixtures, and no damage to agriculture may be caused by its fume. Such conditions are rarely fulfilled at individual mines, and to satisfy them the ore has often to be sent some distance. The question of freight then makes it necessary to reduce the bulk of the ore as much as possible, its richness increasing. Accordingly, the copper content of the material smelted is generally above 10 per cent and sometimes as high as 35 per cent, and the practice of copper smelting has adjusted itself thereto, now taking place to a great extent in reverberatory furnaces instead of the blast furnaces associated with a lower content. Such bulk reduction demands much dressing.

In the case of lead the position is similar, though much lead ore containing 10—15 per cent of the metal is smelted in blast furnaces, much also containing 40 per cent is smelted in reverberatory furnaces, and some above 60 per cent. No closer idea of the lead content acceptable to the smelter can be given, marketing conditions varying greatly. Exceptionally, it has been demanded that the content shall be as high as 70 per cent.

With regard to zinc, retorting being impossible with impure ores, the range in value of material acceptable to the metallurgist is smaller. Though a content as low as 30 per cent may be accepted with oxidized ores the general base for the sale of such ores is 40 per cent of zinc, while that for sulphide ores is 50—60 per cent. Below these percentages proportionate penalties are inflicted; above them premiums are given.

With tin the difficulties and cost of smelting likewise increase with decrease in content, so that the net return from low-grade material is doubly diminished. Though, therefore, a concentrate assaying as low as 55 per cent may be smelted when the smelter is near by, a higher content is demanded when freight is a consideration, and it is more common for dressed ore to assay higher than 60 per cent, and often above 65 per cent.

Another difference between crude and dressed ore is the frequent presence in the former, of mineral impurities, metalliferous or nonmetalliferous, which if present would actively interfere with the success of metallurgical operations. In smelting, one of the essentials is to produce a fusible and light slag. Against this several objectionable substances militate. Alumina, barium, and zinc all render slags pasty, necessitating greater heat to keep them fluid; an excess of silica does the same. Yet alumina is generally present in crude ores, barite a common gangue-mineral, and blende a frequent impurity in other ores.

Should the ore offered for smelting contain such objectionable substances penalties are imposed. With copper sometimes all the insoluble portion of the ore including the silica is penalized; sometimes silica is penalized only when there is an excess beyond that necessary to flux the iron present. Frequently in lead-smelting zinc is penalized when more than 10 per cent is present, since it is not easy to carry away such an amount in the slag or drive it off in the fume. Again, in zinc retorting it is necessary to remove iron so that the substance of the retort may not be attacked, and to limit the lead that the distilled zinc may not be contaminated; accordingly, the iron present is liable to be penalized, and the lead also. Fluorite, likewise, by reason of its fusibility, would harmfully affect the operation of retorting, which for success demands an unfused mass. Yet galena—the lead sulphide—marcasite and pyrite—iron sulphides—and fluorite are common associates of zinc ores.

With tin, copper renders the metal less suitable for plating; iron tends to the formation of the alloy known as hard-head, from which it is difficult to separate the tin. Yet minerals of both these metals are common in tin lodes.

In cyanidation, representative of hydro-metallurgy, complex sulphides, especially when sulphides of copper and antimony are among them, may rob the cyanide solution of much of its efficacy, and the removal of such sulphides is sometimes vital to success.

Finally, a vital difference between crude ore and that suitable for submission to metallurgical operations often exists in the complexity of crude ore by reason of the presence of different valuable metalliferous minerals, successful smelting demanding simplicity in this respect. The sulphides of lead and zinc, for instance, are close associates in deposits, yet only in proportion to the completeness of their separation mechanically, can they be recovered metallurgically. More than that, not only is the unseparated portion of the interfering metal lost, but, whether lead in zinc ore or zinc in lead ore, difficulties in the subsequent smelting are caused. Similarly, lead present in copper ore undergoing smelting is lost, though copper present in a charge of lead ore may to some extent be recovered, in a matte.

Dressing then is necessary: to reduce the ore to a proper size; to remove the gangue to the extent necessary; to remove interfering minerals as completely as possible; and to separate different ore-minerals occurring together.

Advantages of Dressing.—In meeting these technical necessities many economic advantages are obtained, to reap which more fully the dressing operations, started in necessity, often are further developed.

Where smelting follows there is less material to smelt. Leaving out of consideration the costs of converting, refining, etc.—which depend upon the metallic contents eventually recovered, and therefore would not appreciably vary whether dressing formed part of the treatment or the ore were smelted direct—smelting is more costly than dressing. Where dressing, including both milling and concentrating, could be accomplished at an operating cost of 5s. per ton, smelting would cost 15s. per ton; and where a dressing plant would cost £75 per ton treated per day, a smelting plant would cost £150 per ton per day. Considering interest and amortization of these capital sums as part of the cost, the full costs would be about 5s. 9d. and 16s. 6d. respectively. Taking the case that the reduction of bulk by dressing was so low that one-half the material remained to be smelted, there would still be a saving of cost, thus:

	Cost per ton.	Cost. Total.		Cost, Dressing and Smelting per ton of Crude Ore.
1st Case—				
2 tons crude ore directly smelted	16s. 6d.	33s.	=	16s. 6d.
2 tons crude ore dressed	11s. 6d.	..		..
1 ton dressed ore smelted	16s. 6d.	..		..
	28s. 0d.	28s.	=	14s. 0d.

In practice, however, the reduction in bulk is rarely so little as to one-half but generally to about one-tenth, though it varies greatly. At one-tenth the calculation of saving in cost of treatment would be:

	Cost per ton.	Cost. Total.		Cost, Dressing and Smelting per ton of Crude Ore.
2nd Case—				
10 tons crude ore directly smelted	16s. 6d.	165s.	=	16s. 6d.
10 tons crude ore dressed	57s. 6d.	..		..
1 ton dressed ore smelted	16s. 6d.	..		..
	74s. 0d.	74s.	=	7s. 5d.

Or if it be allowed that the smelting of this concentrate would cost more per ton, say twice as much, because of the smaller quantity handled and the greater care necessary with richer material:

	Cost per ton.	Cost. Total.		Cost, Dressing and Smelting per ton of Crude Ore.
10 tons crude ore dressed	57s. 6d.	..		..
1 ton dressed ore smelted	33s. 0d.	..		..
	90s. 6d.	90s. 6d.	=	9s. 1d.

An almost equally important advantage coming from this reduction in bulk is the saving in cost of transportation to the smelter. In greater part base-metal ores are sent away to be smelted and there is freight to pay. If the smelter be near so much the better, freight is not then so important a matter. But where the ore has to be sent hundreds of miles by rail or thousands of miles by sea, freight is a large consideration and not infrequently the largest. Whatever reduction of bulk dressing can accomplish is therefore all to the good.

Furthermore, when in the traverse from mine to smelter a frontier is passed where a tax or duty is imposed, it will generally be found that all ore is rated as of a stated high grade. If then the concentrate does not reach the stated figure the impost is needlessly increased, whereas if it be of still higher grade the impost will in effect be agreeably lessened. Accordingly, it is usual to find that concentrate for shipment to distant smelters is prepared more carefully than that to be smelted near by.

A further advantage of dressing—of secondary importance perhaps but one which greatly facilitates the smooth running of the subsequent smelting operation—is the regular character of the dressed ore. In addition to removing the gangue, passage through the concentrating plant removes also the irregularities characteristic of crude ore. Accordingly, the material which eventually passes to the smelter is of such regular chemical and mechanical character that in the provision of flux the margin of safety may be substantially reduced, and charges are made up more economically.

Per Contra Considerations.—The greatest point against dressing is the relatively large loss of mineral sometimes sustained in the operation. The loss in smelting is comparatively small, being generally about 10 per cent when smelting crude ore and 5 per cent when smelting dressed ore. That sustained in dressing is of the order of 20 per cent. With such greater loss the above enumerated advantages of dressing may be largely offset. Counting loss as a part of cost, and assuming a gross value for the crude ore of 40s. per ton, the two cases previously calculated to show the advantage of dressing may be worked out afresh, thus:

	Cost per ton.			Cost.		Cost; Dressing, Smelting, and Losses per ton of Crude Ore.
	Operating.	Loss.		Total.		
1st Case—						
2 tons smelted @	16s. 6d. +	4s.	=	41s. 0d.	=	20s. 6d.
2 tons dressed @	5s. 9d. +	8s.	=	27s. 6d.		. .
1 ton smelted @	16s. 6d. +	4s.	=	20s. 6d.		. .
				48s. 0d.	=	23s. 0d.
2nd Case—						
10 tons smelted @	16s. 6d. +	4s.	=	205s. 0d.	=	20s. 6d.
10 tons dressed @	5s. 9d. +	8s.	=	137s. 6d.		. .
1 ton smelted @	16s. 6d. +	15s.	=	31s. 6d.		. .
				169s. 0d.	=	17s. 0d.

Leaving out of consideration any question of freight these figures indicate that in the first case it would be better not to dress the ore, and in the second case of some benefit to do so.

It will be realized therefore that the question whether to dress or not would often turn upon the question of freight. Taking a freight of 10s. per ton the two cases would work out as follows :

	Cost per ton. Operating and Loss. Freight.		Cost. Total.		Cost; Operating, Loss, and Freight per ton of Crude Ore.
1st Case—					
	2 tons @ 20s. 6d.+10s.	=	61s. 0d.	=	30s. 6d.
	2 tons @ 13s. 9d. ..	=	27s. 6d.		..
	1 ton @ 20s. 6d.+10s.	=	30s. 6d.		..
			58s. 0d.	=	29s. 0d.
2nd Case—					
	10 tons @ 20s. 6d.+10s.	=	305s. 0d.	=	30s. 6d.
	10 tons @ 13s. 9d. ..	=	137s. 6d.		..
	1 ton @ 31s. 6d.+10s.	=	41s. 6d.		..
			179s. 0d.	=	18s. 0d.

The figures now are entirely in favour of dressing, those of the second case showing a very great advantage in favour of that operation.

The foregoing calculations are given to illustrate the possible effect of operating cost, operating loss, and freight, upon the question of dressing. With crude ore suitable for direct metallurgical treatment these considerations largely determine whether dressing shall be applied or not. More generally, however, crude ore is not possible of such direct treatment, but dressing is a technical necessity.

ORE CHARACTER IN RELATION TO DRESSING

The extent to which dressing is carried as well as the operating cost and the loss in dressing, are largely determined by the character of the ore itself, in respect both to its composition and its texture.

Concerning compositional character, in ore-deposits the primary combinations of most metals are with sulphur to form sulphides, these predominating; with arsenic, tellurium, etc., to form arsenides, tellurides, etc.; with oxygen to form oxides; and in the case of iron, with carbonic acid to form the carbonate. The corresponding primary ores are with minor exceptions amenable to dressing.

The secondary combinations are with much the same elements, but the oxides and carbonates predominate. These secondary ores, in so far as they are the oxidized products of sulphides, arsenides, etc., are, by reason of their friable character, generally unsatisfactory to dress, and sometimes impossible. They are mostly smelted direct.

Compositional character also influences the extent of dressing when it introduces two or more valuable minerals which require separation before metallurgical treatment can reasonably proceed; and again, when it introduces a heavy gangue-mineral, such as barite, fluorite, rhodonite, or garnet, which requires for its removal methods other than those based on density.

Concerning textural character, an ore may be: massive, when, containing little or no gangue, dressing will be simple and limited; intergrown, when, containing much both of mineral and gangue, a moderate amount of dressing will be necessary; disseminated, when, containing but little mineral, and that finely distributed, dressing will be extensive.

In greater detail, intergrown ore may be coarsely aggregated, with mineral and gangue large enough to be picked out separately by the hand when the ore is broken, this representing the most favourable condition; or it may be microscopically aggregated with the mineral and gangue particles so interlocked that the finest comminution is necessary for an adequate release, this representing the least favourable condition; or it may be granularly aggregated, corresponding to an intermediate condition.

When to the complexity of an ore containing two or more valuable minerals in a heavy gangue the further complexity of a finely intergrown texture is added, the greatest demand is made upon dressing, should which fail the deposit presently is worthless. A classic instance of such a demand successfully met is that of Broken Hill, New South Wales, where galena and blende occur with some pyrite in a rhodonite-garnet-calcite-quartz-felspar-fluorite gangue something in the following proportions:

Galena	18 per cent.	7·5 specific gravity.
Pyrite	7·0 ,,	5·0 ,,
Blende	23·9 ,,	4·0 ,,
Rhodonite and garnet . .	9 0 ,,	3·6 ,,
Fluorite	2·5 ,,	3·2 ,,
Calcite	7·0 ,,	2·7 ,,
Quartz	30·0 ,,	2·6 ,,
Felspar	3·5 ,,	2·5 ,,
	99·5 per cent.	

This crude ore contains about 15 per cent of lead, 12 per cent of zinc, and 5 oz. of silver per ton. For a long time, only the galena and a portion of the silver could by any means then available be marketed; beyond this and to allow a quartz tailing to be discarded and a zinc-rhodonite-garnet middle product to be preserved, water concentration could do no more. But with the advent of flotation concentration, which, ignoring density, appealed to surface properties, the zinc became separated from the heavy gangue and in greater part recovered.

The case of Broken Hill is illustrative of what may be expected to be of more frequent occurrence in the future. In the natural process of selection the simple and rich deposits received first attention, and to-day sees already the relatively complex and poor deposits in exploitation; the disseminated deposits are, for instance, a discovery of present-day conditions. It seems likely, therefore, that in the future crude ore will become more and more complex or of lower value, and dressing will be burdened with the increasing work of a greater preparation.

Mineral Properties making Dressing possible.—If the natural association of the various minerals in ore-deposits is responsible for the necessity to dress and the difficulties of dressing, the physical properties of the minerals and sometimes those of the ore make concentration and separation possible. These properties are: specific colour, lustre, general appearance, density, surface energy, magnetic permeability, electric conductivity, etc.

Of the first three properties advantage is taken in hand-picking and sorting. All minerals have distinctive colour and characteristic lustre, by which they largely declare themselves; upon that there is no need to insist. Where, however, these are masked by the sparse distribution of the mineral the ore itself may have a distinctive appearance, as, for instance, is strikingly the case with the "banket" of the Witwatersrand gold-fields, no difficulty being experienced in picking out pieces of this auriferous conglomerate from among the pieces of quartzite broken with it.

The remaining properties are those of which advantage is taken in mechanical dressing.

Of these the most important is *Specific Density*, commonly expressed in terms of Specific Gravity; most metalliferous minerals are of relatively high density. This property, it is true, can be appreciated by the human intelligence, and advantage of it may be taken in hand-picking; the sense of weight in the hand would declare the metalliferous nature without assistance from sight. In mechanical dressing, however, it is not by greater pressure upon a support that higher density usually takes effect, but by a more rapid rate of fall in a resistant medium, generally water, or by the greater inertia of the denser particle. Differences in density are the basis of that most important means of dressing, *Water Concentration*, as well as of *Pneumatic Concentration*.

Next in importance comes *Specific Surface Energy*. As with liquids, each solid, and accordingly each mineral, possesses a surface energy peculiar to itself. This energy, unlike liquid surface-energy, cannot be directly measured, but it manifests itself when a solid is brought to an interface between two fluid phases, say, that between air and water. Then, according to the equilibrium obtaining between the three interfacial energies present, the solid will become wetted and sink, or it will hold back the water and float provided it be fine enough. With sulphide ores crushed to a pulp with water the solid-liquid interfacial energy of the mineral particle is relatively high, or by the addition of suitable agents can be so made, and the particle forming part of a system containing potential energy and being free to move, will break upwards through an air-liquid interface to bring the potential energy to a minimum. With gangue it is the other way; the solid-liquid interfacial energy is relatively low, or can be so made, and the particle finds its condition of stability in the liquid. The dressing operations which in this manner make use of differences in surface energy constitute *Flotation Concentration.*

Next, but no longer of primary importance, come the magnetic properties of minerals. Each mineral has a specific power to concentrate lines of magnetic force within its mass, and thus to develop polarity, that is, to become magnetic; this property is known as its *Magnetic Permeability.* Some minerals, such as magnetite, have a high permeability, that is, they are highly magnetic; others, such as wolframite, are feebly magnetic; while others again, such as cassiterite, are non-magnetic. Differences in magnetic permeability are the basis of *Magnetic Separation.*

Specific Electric Conductivity is made use of in *Electro-static Separation.* Most of the ore-minerals are good conductors, and in consequence quickly become charged and repelled when placed in contact with a charged body. Contrariwise, most of the gangue-minerals are non-conductors, and, becoming charged only at the actual contact, are not so vigorously repelled. Differential paths may thus be forced upon mineral and gangue, and a separation thereby be effected. It is possible that the future may see *Specific Inductive Capacity* a means of separation.

Exceptionally, ready *Fusibility* may be made use of to separate stibnite from its gangue, stibnite being the most fusible of minerals. *Decrepitation* under heat has served to effect the removal of barite from a blende concentrate, the barite upon heating going to pieces, separation by sizing then being possible. *Friability* of the mineral will sometimes result in greater pulverization of the valuable portion than of the stony portion of the ore, making separation again possible by sizing. The fibrous mineral asbestos, on the other hand, resists pulverization while the containing rock yields completely; the asbestos becoming fluffy may be sucked into an air chamber and thus recovered.

Fracture, as expressed by the shape into which a mineral breaks, though it may affect the separation of metalliferous minerals by other means, does not itself afford any basis for a separation. In the removal of shale from coal, however, advantage is sometimes taken of the fact that the latter breaks in equidimensional pieces and the shale in flat pieces, by a screen set to pass pieces lying flat.

Coal and shale differ also in their *Coefficients of Friction,* and in that difference provide a further basis for their separation, of which some advantage is taken.

Hardness affects the work of comminution but is not itself a basis of separation. Similarly, *Texture and Aggregation* affect the work of separation but do not themselves afford bases of separation.

CHAPTER II

WASHING AND SORTING

WASHING

THE first essential in separation is the release of the mineral from its attachment to gangue. Generally the ore must be crushed. Sometimes, however, release can be effected by disintegration with water, that is by washing.

Among iron ores, limonite often occurs as nodules embedded in clay, while hæmatite occasionally occurs distributed through sandy material. Among lead ores, it is common to find masses of galena associated with residual clays; a similar occurrence of copper ores is not uncommon. The oxidized ores of zinc are likewise largely associated with clay and other impurities resulting from weathering. Manganese ores also are largely found in surface deposits mixed with a good deal of clay, etc. Finally, the gravel deposits of gold, tin, platinum, etc., contain their valuable constituents in an overwhelming mass of gravel, sand, clay, loam, etc. With all these and similar ores, release and sometimes final separation may be effected by the disintegrating and carrying power of water, assisted by such mechanical means as may be found necessary.

Though such release may represent washing in its greatest development, washing has a further well-defined purpose in cleaning ordinary crude-ore preparatory to the subsequent operations of sorting and crushing.

Apart from the shallow deposits mentioned above, steep-standing deposits often have in their upper portions a pronounced weathered zone characterized by much clay. In depth also such deposits are frequently associated with deep-seated clayey alterations of the country rock, propylitization and kaolinization, for instance; or they may be associated with shear and attrition zones containing excessively-comminuted material. Moreover, the clay-lined walls of many such deposits are characteristic.

Even where no natural clay be present much fine material resulting from blasting settles everywhere, but largely upon the broken rock, to which, with the aid of the moisture in the atmosphere underground, it clings tenaciously, masking the material it covers.

By washing not only do hard ores become the better displayed for sorting, but at the same time relieved from material which in crushing might cushion the blow or clog the discharge.

Washing Appliances

Log Washer, or *Trough Washer.*—This washer consists of a tank or trough about 18—25 ft. long, 4—7 ft. wide, and 4 ft. deep at the lower end, the bottom rising about 1 in 12 so that the upper end is just above water (Fig. 1). In this tank, and supported in bearings at either end, are laid two cast-iron or steel 'logs' or shafts driven by gearing to rotate some 8—16 revolutions per minute in opposite directions. These logs have upon them a large number of stiff chilled-iron blades, so obliquely set that upon revolution they exert an upward conveying effect upon the ore introduced near the lower end. In this upward movement the ore meets water arriving through the bottom and flowing downward. At the same time the blades, which in their revolution come close to the bottom, stir the ore and crumble the larger pieces, giving opportunity for the descending water to carry the lighter non-metalliferous portion over the lower end, while the mineral continues its forced way upward, to be discharged at the upper end (Fig. 2).

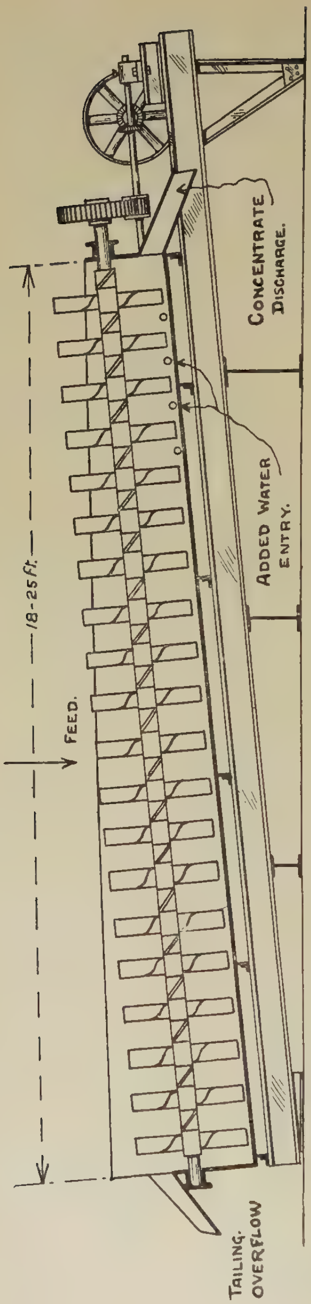


Fig. 1.

Log Washer.—Diagrammatic Section. Only one log is shown. Two blades at 180° make a unit screw, the intervening angles of 90° being taken by another two blades upon the next screw, there being altogether 40 or 50 such screws. In the illustration alternate screws are shown with their blades vertical and horizontal, respectively (p. 14).

The material fed into these log washers is generally the undersize of a screen with about 2½ in. aperture. If desired, the impoverished overflow from one washer may after settlement be passed in a thickened condition through a second and smaller washer, the dimension and action of which are better suited to the treatment of finer material (Fig. 3).

These washers have received their greatest application upon the sandy hæmatites of the Mesabi iron range, Minnesota. They are also largely used in Florida to clean 'phosphate,' and have been used elsewhere to clean oxidized zinc ores, etc. They have a large capacity, a washer of standard size being generally capable of treating 25—100 tons per hour, the exact figure depending upon the amount of clay or sand present. The earlier forms of this appliance had usually but one log, and the capacity was consequently much lower.

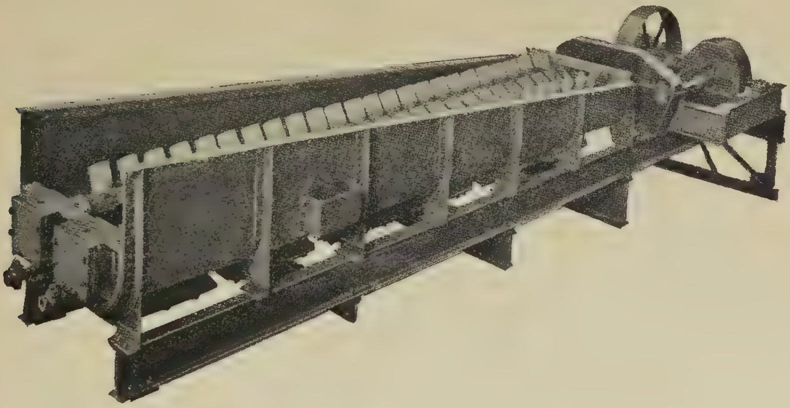


FIG. 2.

Log Washer.—Perspective View (Allis Chalmers). Two logs are shown, their lower ends being secured in substantial enclosed bearings over which the discard flows. Along the length are six supports to the trough. Between each two supports is a pocket or hutch in the trough bottom, these pockets being connected with a water service (p. 14).

Log washers are as simple in design as they are effective in operation. They require little power, generally less than 5 h.p., and their operating costs are low. On the Mesabi they deliver a cleaned product which contains 50—60 per cent of iron, discarding at the same time one-half to twice as much impoverished material containing about 12 per cent of iron. The consumption of added water is 0·5—1 ton per ton of ore treated.

Drum Washer.—The log washer is chiefly known in America. In Europe the washing drum serves a similar purpose. Ordinarily, this

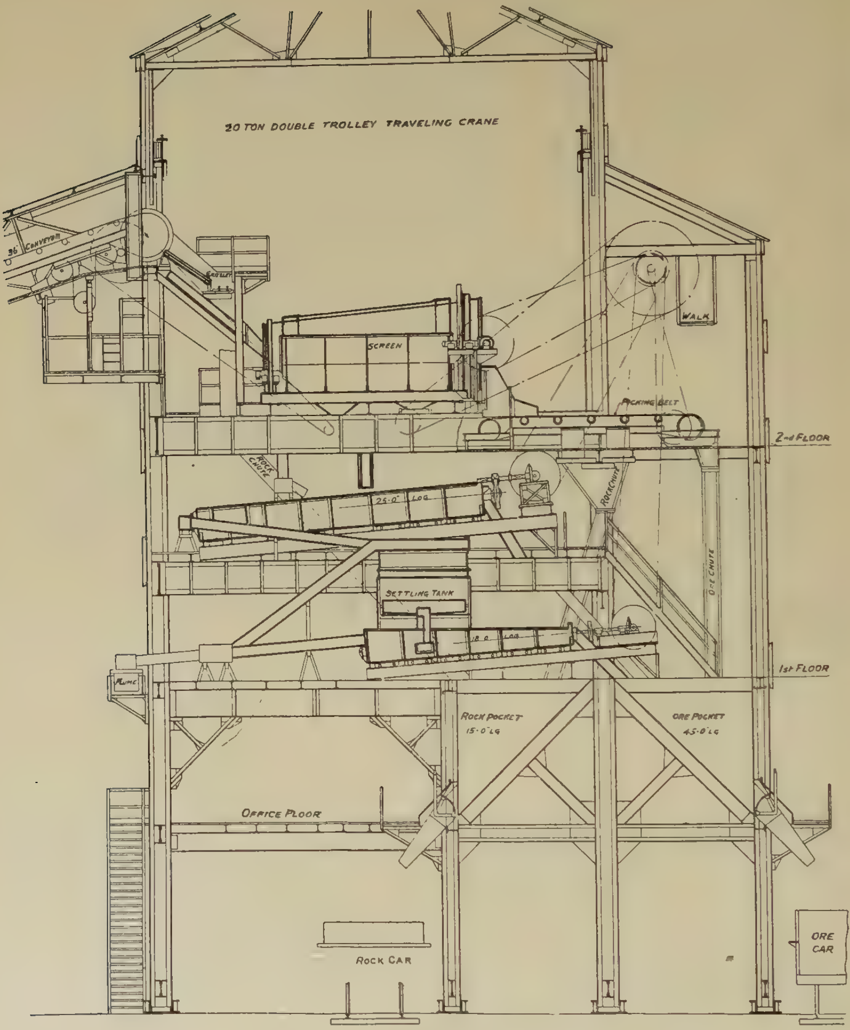


FIG. 3.

Washing Plant on the Mesabi Range.—Side Elevation (Allis Chalmers). This illustration shows the crude ore delivered on to a screen (grizzly), the oversize pieces rolling to a platform where they are broken by hammers, the smaller pieces passing into a conical trommel set within a housing. The oversize from this screen passes to a picking belt, where waste is picked out and dropped down a chute, while the mineral passes on, to be delivered over the end. The undersize collected in the housing passes down to the first washer 25 feet in length, which delivers cleaned ore at the upper end and an overflow of impoverished material at the deep end. This latter passes to a settling tank from whence relatively clear water is taken off at the top, while thick pulp issues at the bottom to drop into a smaller washer below, the overflow from which is the final discard. It will be noticed that whereas there are entries for water along the whole bottom of the smaller washer, these entries only extend half-way down the bottom of the larger washer. In the smaller washer there is a greater proportion of waste to remove (pp. 15, 20).

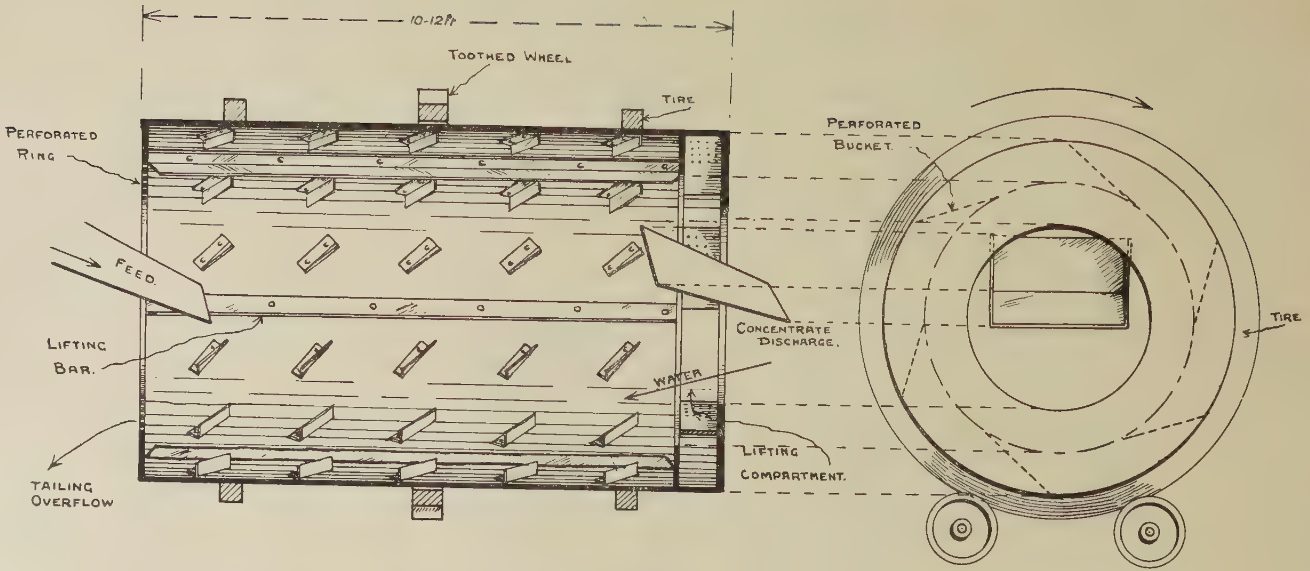


Fig. 4.

Drum Washer.—Diagrammatic Views. In the longitudinal section, feed, concentrate discharge, and tailing overflow, are shown. The rim which determines the overflow is shallower than that over which the mineral is discharged, so that a gradient between the two ends is established. That rim also determines the depth of the water and material within the drum. It will be noticed that the buckets which lift the cleaned mineral are perforated, so that water is not raised (p. 18).

appliance is a large steel-plate cylinder about 6 ft. in diameter and 10 ft. in length, supported at each end on rollers, and rotated some 40 revolutions per minute by the engagement of a pinion with a toothed wheel medially disposed around its body (Fig. 4). The material to be washed is fed through a large central opening at one end, the cleaned material being lifted by perforated scoops or buckets to a similar discharge-opening at the other end. Progression towards the discharge is sometimes assisted by plates bolted obliquely to the shell to give a forward impulse during revolution; discharge of the water at that end is prevented by providing facilities at the feed end (Fig. 5).

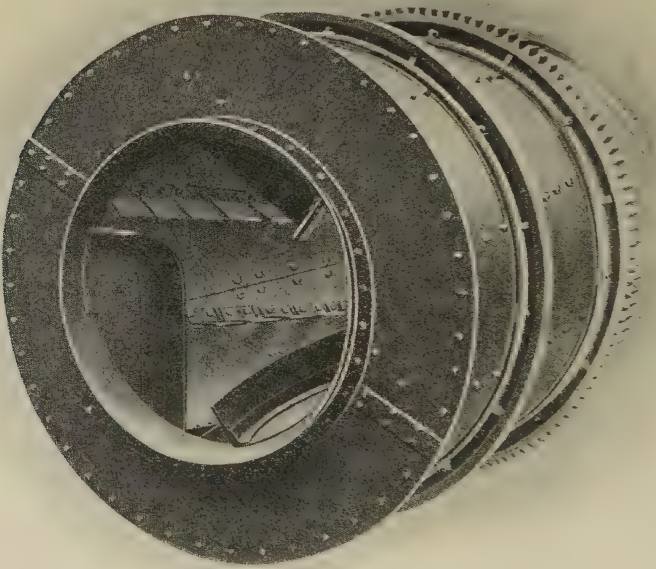


Fig. 5.

Drum Washer.—Perspective View. Looking into the drum through the discharge end, rows of lifting ribs are seen disposed longitudinally and at regular intervals around the interior. A novel type of discharge bucket is shown, just capable of lifting over the rim; of these buckets there are three. The toothed wheel around the body is seen medially placed between the two supporting tires. The overflow of the muddy suspension takes place at the far end (p. 18).

The ore is generally fed dry, water being added through the discharge end to the extent of about 1 to 2 tons of water per ton of material. This water, while serving its disintegrating purpose, passes through the drum in a direction opposite to that of the mineral, eventually overflowing

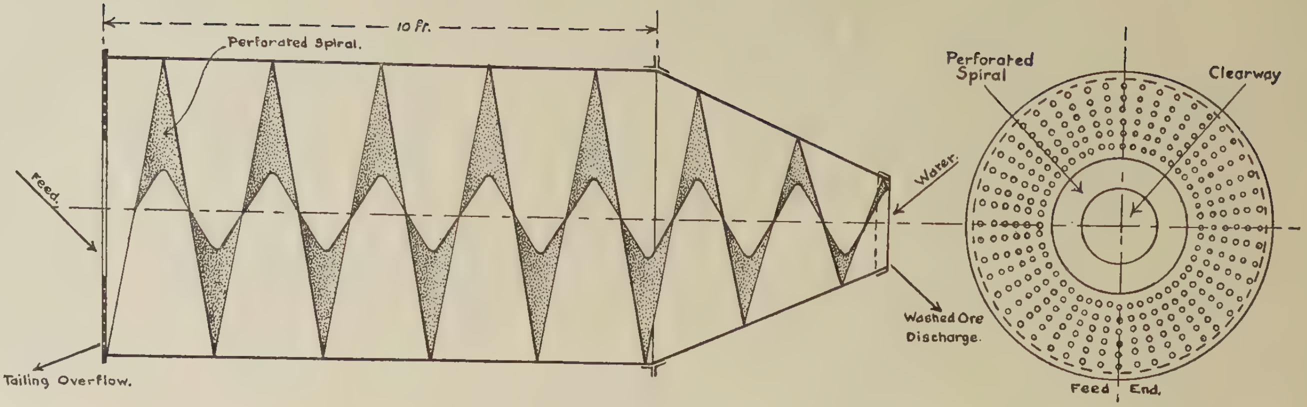


FIG. 6.

Spiral-flighted Drum Washer.—Diagram. The spiral flight on the interior is of perforated plate throughout, so that the muddy suspension can make its own gradient back towards the feed end. The shallow depth of the rim over which this suspension overflows will be noticed; there will accordingly be but little depth of water at that end. The view of the feed end shows the perforated plate through which suspension leaves the washer. The spiral plate reaches about two-thirds of the way from periphery to the centre, leaving a central clearway through the drum (p. 20).

through a perforated ring at the feed end, carrying with it the material brought into suspension. The rim over which this overflow takes place determines the depth of water in the drum. To assist in the disintegration, ribs of angle-iron bolted to the shell at regular intervals around the periphery raise the ore during revolution to drop it again as they rise above the centre.

These washers have been largely used on the Continent for cleaning iron ores, lead ores, and manganese ores, the granular sorts of these last ores being easily brought up to 85 per cent of manganese dioxide. They have a large capacity, some 10—20 tons per hour, upon material which is not too clayey ; the power required is not great, something of the order of 5 h.p. per washer ; and the operating cost is not high.

Where the ore is very clayey and a longer treatment necessary, the drum, instead of having longitudinal ribs upon its interior, may have helical flights of perforated steel plate projecting towards the centre (Fig. 6). During revolution, progression towards the discharge is then positive but slow, actual discharge being finally effected through a conical discharge end, up which the flights continue. Such drums are used in the Bilbao field to enrich fragmentary limonite deposits. There they are 5 ft. in diameter and 10 ft. in length, excluding the cone ; are driven 90 r.p.m. and require 12 h.p. ; have a capacity of 10 tons per hour, and require 2 tons of water per ton of ore to be washed.

Instead of being horizontal, such flighted drums may be laid at an inclination, when there would be no need for a conical end to hold back the water while facilitating the discharge.

In another design of drum washer a set of paddles arranged around an independently-driven central shaft is rotated rapidly within the drum and in an opposite direction to it. The material, lifted by the longitudinal ribs bolted to the shell, is then struck in its fall by the rapidly rotating paddles, the most clinging attachment being thereby broken and the most tenacious clay puddled.

Washing Trommel.—Revolving trommels, these being drum-shaped screens, constitute an excellent means for cleaning ore (Fig. 3). The rough-and-tumble passage down the trommel, taking place as it does under the searching action of water, provides all the opportunity necessary for washing preparatory to sorting, and such trommels are largely used for that purpose. The holes in the trommel are then of a diameter to pass only the unsortable fines.

Either the cylindrical or conical trommel may be used, the latter having the advantage that with axis horizontal it is more conveniently driven, while the cylindrical trommel, though it requires to be set with its axis somewhat inclined, is more simply constructed (p. 235). The trommels used for washing are large; having to bear the passage of large pieces they must also be robust. They are generally about 12—15 ft. in length and 3—5 ft. in diameter, though for heavy work and large quantities, as for instance with soft iron ores or when dredging alluvial deposits, these dimensions are greatly exceeded. They make about fifteen revolutions per minute. The better to support their great weight they are generally carried on rollers, and not upon arms around a central shaft. Their capacity is large, a trommel of the dimensions given being capable of washing about 20 tons per hour. Discharge of the undersize or of the puddled material being directly downward through the walls and not at the end, the amount of water necessary is somewhat less than with washing drums, and generally about 0·5—1 ton of water per ton of ore.

Washing on fixed Screens, Grizzlies.—Where fixed screening-appliances are used to separate the unsortable fines, the coarse material is often cleaned for subsequent sorting by the play of water jets upon it, either as it moves down the screen or as it lies collected at the bottom (Fig. 9). Such fixed screens, be it remarked, must be set at a comparatively high angle, generally above 35°; they usually deliver directly and freely on to the sorting appliance, but sometimes this delivery is regulated by a gate, making it possible to keep the material on the screen, and consequently under the play of the water, longer than otherwise.

Washing by Pressure Water.—Water under pressure has sometimes been advantageously used in washing. A jet of water issuing under a pressure of, say, 50 lb. has great disintegrating force, and such jets have sometimes been the most effective way of removing much clay. For their successful application the ore must either be brought to a discharge gate facing the jet, or the jet may be directed into an iron cylinder open in front for the entry of the jet and for discharge, and on the top for the feed. More water is required by this means, but since there are no moving parts the installation is simple. In hydraulic mining such jets are played directly upon the deposit.

Puddling Machine.—The early appliances for washing ores consisted of a circular trough around which knives or harrows, fixed to or dragging from radial arms, were drawn by horses. This trough was generally cut in the ground and lined with stone or wood; into it the ore was thrown and water was turned. Such contrivances, known as puddlers, were, however, only suited to intermittent and consequently small-scale work; they cannot be said to have any place in present-day practice. The clay was carried away in suspension, the cleaned ore remaining to be lifted out when the washing of a charge was complete (Fig. 7).

At the Kimberley diamond mines an improved form of puddling machine for washing the decomposed blue ground is used (Fig. 8), which consists of an annular iron trough or pan, of 15 ft. diameter outside and 8 ft. diameter inside, set on pedestals. At the outside rim—where the material is fed with a proper amount of water to form a pulp of thick consistency—the depth of this pan or mixer is about 18 inches, and at the inside about 10 inches. Around the trough so formed a system of paddles attached to a central spindle rotates about 10 revolutions per minute, the paddles being so set that in addition to stirring the material they tend to force the solid portion of the pulp outwards, while the resultant muddy suspension overflows inwards. The paddles accordingly act against the flow. The percentage of heavy material finally remaining is about 2—3 per cent of the weight of material originally fed. This concentrate accumulates till the pan is considered to be reasonably full—reasonably, having regard to the fear of casting out diamonds—when the feed is stopped, and the accumulation removed through a slot opened in the pan bottom, to

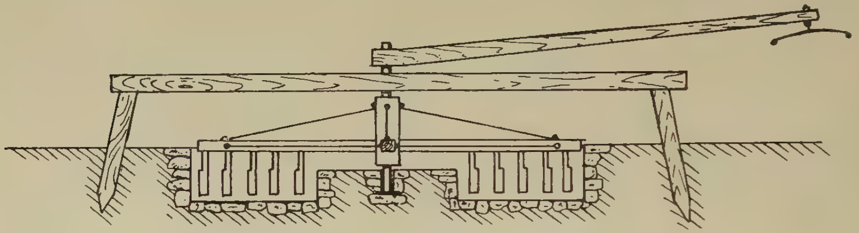


FIG. 7.

Puddler.—Primitive Type (p. 21).

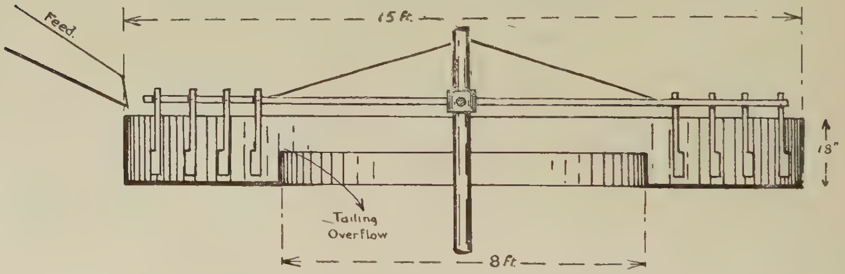


FIG. 8.

Puddler, or Mixer.—Kimberley diamond mines, South Africa (p. 22).

which slot all is brought by a plough lowered from one of the rotating arms.

It should be remarked that, ordinarily, puddling is taken to be the operation by which fine, dry or stiff material is mixed with water to form a uniform sludge, as when forming mortar, or bringing dumped residues again into the condition of pulp for retreatment. Puddling machines and drum washers may be used equally well for such a purpose.

SORTING, HAND-PICKING

Sorting or hand-picking is the separation of different constituents or classes of ore by hand, taking advantage of those mineral characteristics appreciated by the eye, namely colour, lustre, and appearance. It is, in short, the hand-dressing of ore, distinct from the more extensive mechanical dressing.

The aims of this operation are: where possible, to pick out pieces of ore clean enough for direct metallurgical treatment, these thus escaping the loss and avoiding the expense of further dressing; to sort out pieces of gangue so obviously worthless that any further work upon them would be futile; and to pick out and keep separate those different minerals of a complex ore, or those different classes of a simple ore, which the ordinary milling operations could only result in more intimately mixing.

Concerning the first aim; galena, blende, chalcopyrite, native silver, etc., frequently occur in their respective deposits in pieces large enough to be hand-picked. The second aim can almost invariably be realized, since only with very low-grade ore can waste not be sorted with advantage; ordinarily, in addition to that picked out underground, a percentage amounting often to twenty and more, may be sorted on surface. The third aim is particularly possible of realization with complex ores containing two or three such minerals as galena, blende, pyrite, and chalcopyrite, in a coarse state of aggregation (Figs. 235, 284); but also with massive simple ores of copper and iron.

In addition, when sorting, the opportunity is taken to remove such minerals and foreign materials as would likely interfere with the smooth running of the subsequent operations, namely: pieces of steel which might cause breakage in the crushing equipment; wood shavings and chips which might cause choking of pipes and spigots, or the blinding of screens; and occasional pieces of such minerals as might interfere with subsequent metallurgical treatment, pieces of stibnite from a gold ore being prepared for cyanidation, for instance. The very gratifying effect of these apparently minor attentions must be credited to the opportunities provided by the sorting equipment.

It will be apparent that hand separation might under certain circumstances be the only separation. Such circumstances would obtain where the mine was small or in its early stages, or where the ore was massive and required a division into such classes as the human intelligence alone could make. In the first case, the lump ore would be broken by sledge hammers —weighing about 10 lb. and worked two-handed—when opportunity would be taken to pick out the pieces of mineral and to discard those of barren gangue (Fig. 284). This operation of *sledging* would be followed by a similar one with a spall hammer weighing about 6 lb., when a further separation would be made. Following *spalling* would come the operation of *cobbing*, which consists in using a chisel-edged hammer — weighing about 3 lb. and worked single-handed—deliberately to break ore from waste. All these three minor operations go to make the complete hand separation, though the term cobbing is sometimes used for the whole operation ; hand-picked ore is, for instance, often known as cobbed ore.

Hand separation, though satisfactory upon large material, is so expensive when applied to smalls that it cannot be continued upon material less than 0·5 in. in size. Yet, though hand-breaking produces relatively little fine material, a large proportion of the ore would quickly reach that limit of size, and so escape beneficiation were only hand-dressing applied. Moreover, the capacity of hand labour to cob ore is very limited, not reaching more than about 1—2 tons per man per shift, depending upon the hardness. Where, therefore, the ore is hard or the quantity great, mechanical breaking must be called in, employing which, a larger amount of fine unsortable material is made and mechanical separation becomes more than ever the rational procedure. With such mechanical means once installed, hand separation becomes limited to the early stage of dressing—which is its ordinary position in present-day practice—and is not continued down to the same small size as before. Often, indeed, it is entirely superseded by mechanical dressing.

Exceptionally, where the ore is massive and sorting would hardly be justified by the small amount of waste present, it may be desired to keep different classes of ore separate ; hand-dressing then would still be an important and extensive operation. At the Rio Tinto copper mines, Spain, for instance, smelting ore, leaching ore, sulphur ore, and quartzose ore, are separated by hand, while at the Gellivara iron mines, Sweden, several classes of ore are separated according to the sorter's estimate of their phosphorus content.

Sorting Appliances

Sorting or picking can to a limited extent be done at the foot of the grizzly on to which the ore coming from the mine is first delivered. The oversize from this grizzly then falls upon an iron-plated floor where it is cleaned by water jets, the larger lumps broken, and opportunity given for discarding waste. If, however, this grizzly be small in width, as those situated in the head-frame usually are, sorting becomes intermittent, consequent upon the intermittent arrival of the ore. Moreover, as ore usually arrives both day and night, some sorting must of necessity be done by night, whereas it is best done by day only, since artificial light is often inadequate for sharp discrimination. Sorting under such conditions is very inefficiently performed and of doubtful advantage.

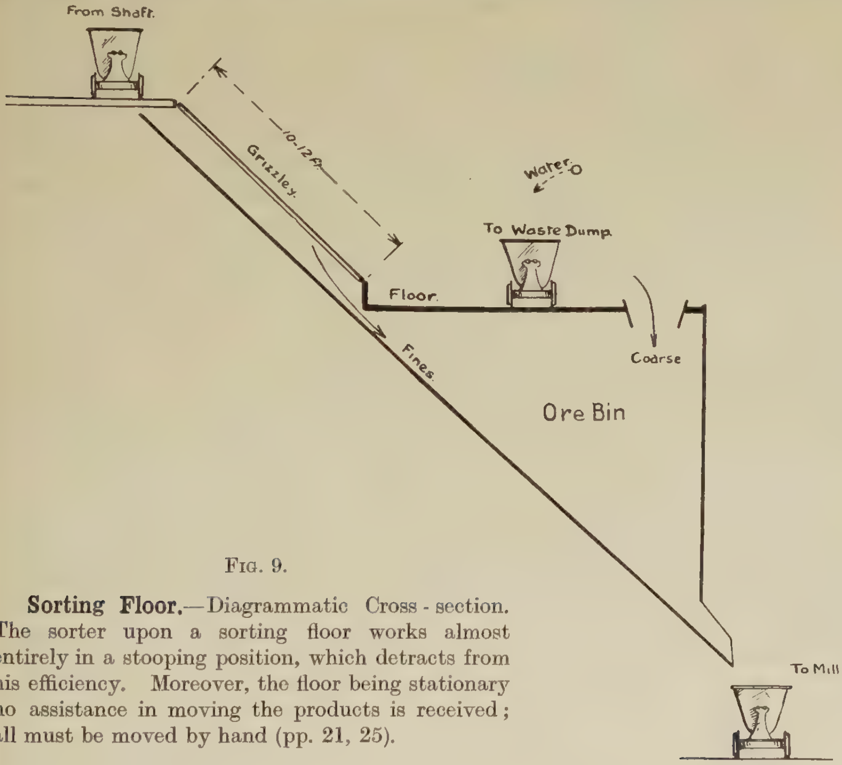


Fig. 9.

Sorting Floor.—Diagrammatic Cross-section. The sorter upon a sorting floor works almost entirely in a stooping position, which detracts from his efficiency. Moreover, the floor being stationary no assistance in moving the products is received; all must be moved by hand (pp. 21, 25).

Sorting Floors.—The same general design as that described above, when developed to provide the necessary facilities for the operation, becomes a sorting floor (Fig. 9). The grizzly is then increased by placing several sections side by side, the iron-plated floor at the bottom being extended similarly. Such a floor is served by a track running along the top of the grizzly, permitting the ore to be dumped where the floor happens to be free; moreover, the ore arriving from the mine may be tipped into a bin to be drawn afterwards during the day-time only, no sorting then being necessary at night. The grizzly bars, set at an angle of about 45°, are about 12 ft. in length and $1\frac{1}{2}$ in. apart. At their lower end is a drop of one foot or so on to the iron plates, this drop giving opportunity for the oversize to collect, the undersize passing directly to a bin beneath. This collected oversize is then pulled apart with rakes and cleaned by the play of water jets, while the large lumps are sledged. Picking is then undertaken, the different finished products, clean ore or waste, being lifted into trucks running upon the floor, while the mixed or milling ore remaining on the floor—this being generally the greater bulk—is shovelled into the mill-bin. The water which runs off from such a floor is generally led to a tank for the fine solids to settle, this material being periodically removed and delivered at some point lower down the treatment.

Such floors are simple and efficient, though with so little mechanical assistance they are expensive in labour. They have been used extensively on the Witwatersrand for sorting waste from gold ore, and at Rio Tinto, as mentioned above, for separating the crude ore into several different classes.

Sorting Tray.—The sorting tray is an iron-lined tray placed at the bottom of the head-frame grizzly, or immediately following a breaker. Suspended by rods, ropes, or chains, this tray lies at an angle of about 10° from the horizontal, and is free to take a backward motion against gravity and a forward motion with gravity (Fig. 10). This reciprocating motion is imparted either by an eccentric—when the momentum of the particle on the movement downward works with gravity to secure a progression of the material—or by a cam which releases the tray at the end of the backward stroke, the tray then falling forward to be arrested against a stop while the ore moves onward by reason of its acquired momentum. The amplitude of the reciprocation varies from 2 in. to 8 in., and the speed from 80 to 250 reciprocations per minute, the smaller amplitude going with the greater speed.

The length of such a tray may be anything from 10 to 25 ft., and the width from 3 to 6 ft., depending upon the width of the grizzly. The length largely depends upon the space available, and, when these trays are used to convey the coarse material to or from the breakers, also upon the desired distance of conveyance.

Sorting is performed as the material passes down the tray, water being played upon it during passage. When placed below the head-frame grizzly, the irregular arrival and heaping of the material at the head of the tray places this part at a disadvantage for sorting, while night sorting must also be done if ore is drawn during the night. When placed below a breaker, the circumstances are considerably better, but the appliance is still far from ideal.

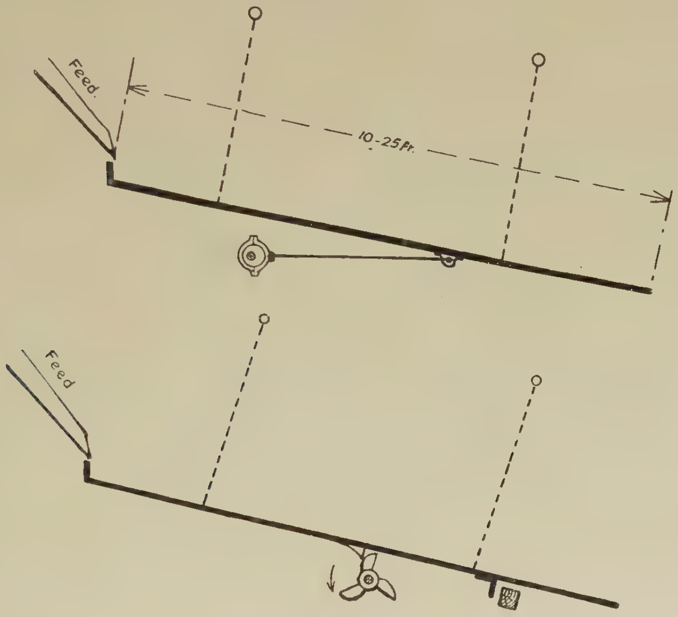


Fig. 10.

Sorting Trays.—Diagrammatic Longitudinal Sections. Progression of the material is brought about by repeated shakes. Such trays can accordingly be used as feeders, for instance, to breakers. They are convenient in that a man at work can stand up, but being heavy and burdened with ore, they strain themselves and their support (p. 26).

Sorting Tables.—Sorting tables may be stationary or revolving. They possess the advantage over floors that the material to be sorted is displayed

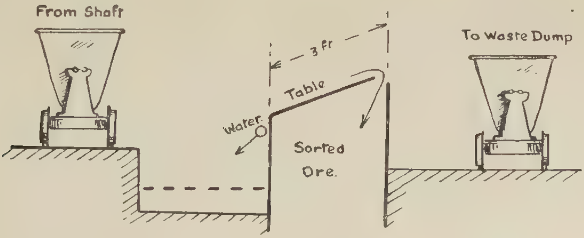


Fig. 11.

Stationary Sorting Table.—Diagrammatic Cross-section. Tables of this type, but more simple, are also used for cobbing, the iron-plated top making a good anvil for the hammer-blow (p. 27).

at a convenient height. The stationary table is the simpler appliance, but one necessitating a good deal of handling and of limited capacity. It is generally a long table covered with iron plate and inclined slightly to one side (Fig. 11). At the foot on the lower side is a trough having a false bottom of perforated plate. The screened ore, arriving in trucks along a track running on the other side of the trough, is tipped into this trough where it is cleaned by water. When clean it is shovelled on to the table, the waste then being picked out and thrown behind into a truck, while the ore is drawn by a scraper to drop into a bin below. If pieces of rich ore can be picked out separately, these are dropped into special baskets or pockets. The actual sorting is done from the far side of the table, the width of which is about 3 feet. The length of the table determines the number of operators, and consequently the capacity; one man is usually required for every 4 ft. or so of length.

Another design of stationary table is octagonal in shape, the crude ore arriving down a central chute. Sorting is then performed as the material is drawn towards the periphery by rakes under a spray of water, an appropriate number of operators participating.

The revolving table represents a considerable advance both in convenience and capacity. It is the stationary table bent round a circle and made capable of rotation around a vertical axis (Fig. 12). The outside diameter of the annular sorting space so provided varies from 15 to 25 ft. When the operators stand outside only, which is the more usual practice, the sorting width is from 24 to 30 in.; but when they stand both inside and outside, as they sometimes do with larger tables, the width is proportionately greater. The surface is not usually quite horizontal but slopes towards the operator to permit any water to drain into a launder. Where sorting is done from the inside as well as from the outside, the table has a double slope. If a flat table is used, there are holes in the plated surface down which the water may drain.

The tables of smaller diameter are generally supported upon an inverted umbrella-frame radiating from an upright shaft at the centre, through which shaft rotation is communicated. Tables somewhat larger may be suspended like a merry-go-round, from above. Those still larger, having a circular track fixed to the reverse surface, are supported upon rollers placed at regular intervals around the circle, rotation being communicated by a driving pinion engaging a peripheral rack (Fig. 13). Whether large or small, the speed at which sorting tables are run is determined by the rate at which sorting can be done, and a peripheral speed of 20—30 ft. per minute is common. At that speed one operator can overlook about 3—5 ft. of the circumference.

The ore usually delivered to the table is the oversize after a first screening

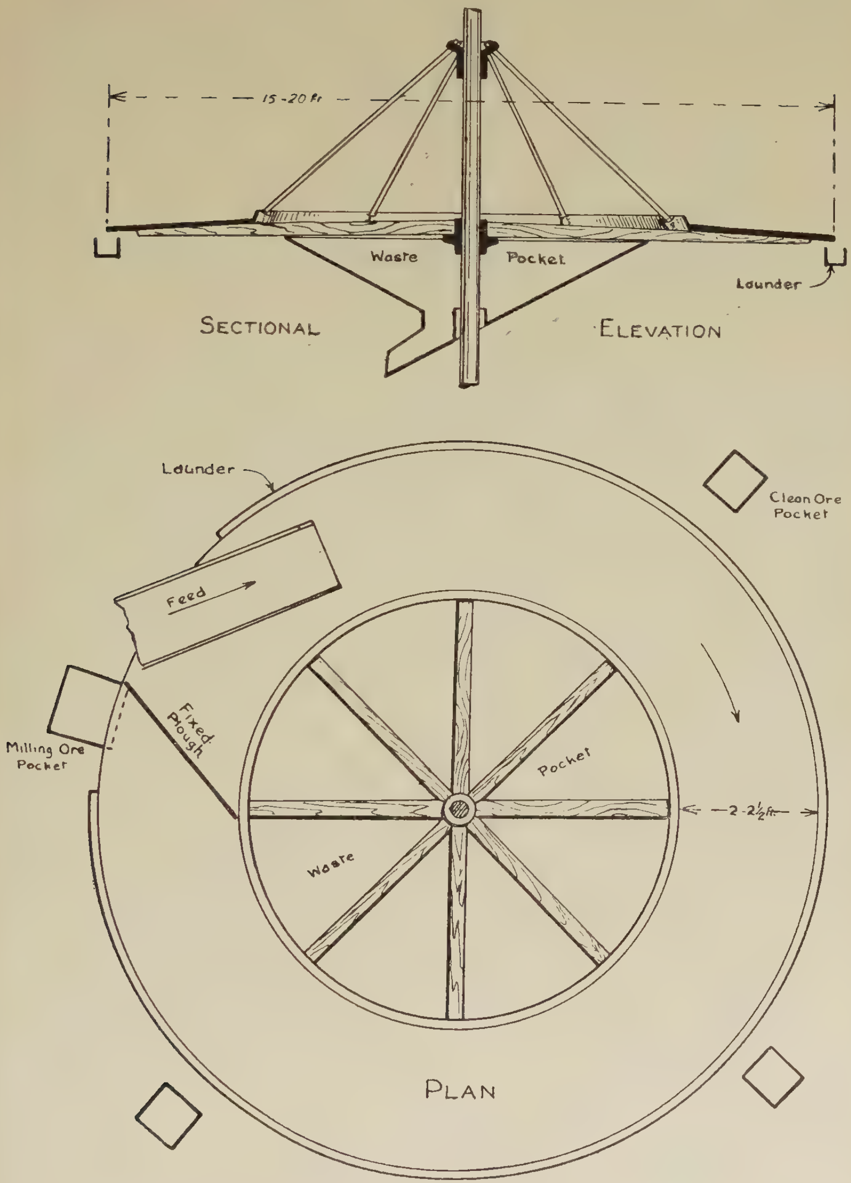


Fig. 12.

Revolving Sorting Table.—Diagrammatic Sectional Elevation and Plan. The shaft-supported type illustrated is the most simple and common, but does not permit inside service as do the other types (p. 28).

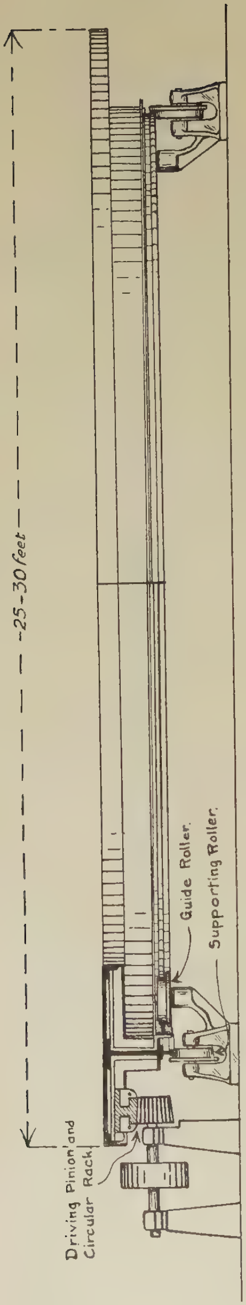


FIG. 13.

Roller-supported Sorting Table.—These tables are firmer, and the movement more steady than with shaft-supported, centrally-driven tables of the same diameter. They are usually worked from both the outside and the inside, communication with the inside being by a bridge spanning the annular width (p. 28).

has separated the unsortable fines. Generally this oversize has been machine broken before delivery to the table, but sometimes, except for the sledging of larger lumps, it is delivered unbroken. Where machine breaking has been applied, the broken product may have received a second screening before being sorted, but this practice is not general.

The ore arrives on the table either directly from a grizzly or trommel, or from a breaker. Before arrival it has been washed or wetted to the extent necessary. Passing in front of the operators the clean ore and the waste, according to the particular practice followed, are picked out and placed in separate pockets, while the mill ore, which generally constitutes the great bulk, passes on to be eventually ploughed off into a bin. Since each operator must be able easily to place the picked material into its proper pocket, there may be quite a number of pockets around the periphery, though those for each particular material would lead into one bin. Tables using both the outside and inside rims have sometimes a projecting rim at the crown, making a double table on which two separate materials may be sorted concurrently, the larger from the outside and the smaller from the inside.

Where only one separation is made, namely, waste from milling ore, or clean ore from milling ore, simplification of bin construction is obtained when the table width is divided between two decks, the upper, that farther from the operator, serving as a shelf upon which the material picked out may be placed. This material can then, likewise, be ploughed off into its proper bin, the bin arrangements being thereby much simplified. Such a table is described as a double-decked table. With some larger tables served from the inside as well as from the outside a raised central portion is sometimes made a common shelf for the waste from both sides.

The capacity of these sorting tables is large and proportional to their width, that of a 30 in. table being about 40 tons per hour, provided the feed be continuous and regular. The length around a table determines the completeness of the sorting accomplished, the larger the circle the greater the number of possible operators and the more thorough the work. If more than one material is picked out, different operators may devote their attention to different products.

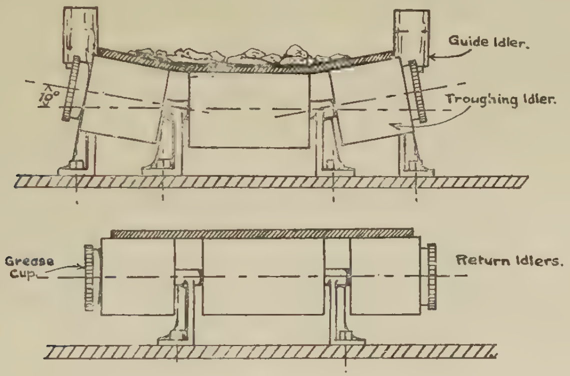


Fig. 14.

Rubber Sorting Belt.—This belt runs upon idlers, those supporting the upper half being set at an angle to trough the belt sufficiently but not excessively; in addition, special guide-idlers spaced at greater intervals keep the belt properly to its course. The idlers supporting the return half of the belt lie flat.

A sorting belt differs from a conveying belt, in its shallower trough; in its more frequent support upon idlers, these occurring roughly every 3—4 feet instead of 4—5 feet; and in its slower speed, 30—40 feet per minute against 150 feet per minute at least (p. 31).

Sorting Belts.—If it be required that the sorting appliance deliver the milling ore without transporting it, the sorting table possesses advantages; but if it be desired that at the same time the ore shall be carried forward or raised, the sorting belt is the appliance to install. This belt is an ordinary conveyer belt having a rubber surface on a canvas backing (Fig. 14). Its width is usually from 24 to 36 in., and its length dependent upon the distance to be traversed. It runs at a speed of about 30 ft. per minute, differing in this respect from the conveyer belt which runs many times faster. In order to contain the material, the belt is troughed upon idle rollers ; the material at the centre is in consequence not so well displayed. The shallow depth at the sides also affects the capacity, and a belt has in consequence a somewhat smaller capacity than a table of the same width and running at the same speed. To minimise these disadvantages the belt is not troughed so deeply as for conveyance pure and simple. With sorting belts also, the disposition of pockets at intervals along the length makes collection of the material into bins more difficult, though this difficulty may be surmounted by having a separate belt to carry away and deposit in one place the material picked out (Fig. 16).

Belts have, however, some compensating advantages. Instead of being set downwards like the shaking tray, or horizontally like the table, they may, within the limit of about 15°, be inclined upwards, height being thereby gained. This upward inclination also permits entrained water to drain away ; washing may even be performed on the belt by playing water on the first portion. The belt is also in many cases a conveyor, and any such service must be recognized as one of its advantages. In obtaining these advantages more power is required than with tables.

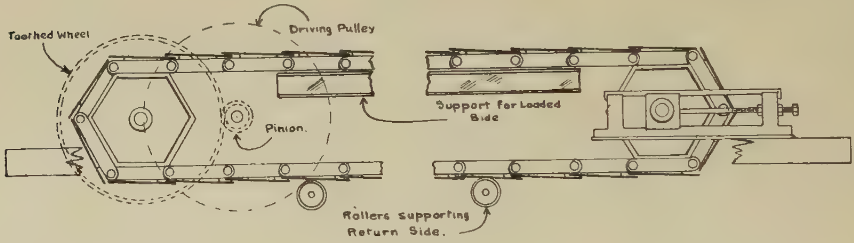


Fig. 15.

Steel-plate Picking Belt.—Diagram (p. 32).

Instead of rubber belts, a string of articulated metal trays passing around polygonal sprockets, is sometimes used where the large size or abrasive character of the ore would make rubber too expensive (Fig. 15). Such belts run at a slower speed than rubber belts and consequently for the same width have a smaller capacity ; even then wear and repair are considerable. They are largely used on collieries chiefly because being flat and wide the material is well displayed.

As with tables, the operators may stand on one side or on both sides, the latter procedure justifying a wider belt ; the capacity again depends upon the width and the speed of running, while the completeness of sorting depends upon the length. Exceptionally, at some collieries, and with a wide steel-belt, the operators stand upon the belt.

Résumé.—The extent to which sorting is carried depends largely upon the quality and the aggregation of the ore.

If the ore is rich there will be some pieces of high value and others containing practically nothing. With this wide range in value there will usually be an equally pronounced difference in appearance, and more extensive sorting becomes possible; moreover, rich ore-bodies are generally small and in consequence more waste is broken with them. On the other hand, when the ore is poor it is usually less distinguishable from the waste, in addition to which, poor ore is generally associated with a large ore-body and the proportion of waste unavoidably broken with it is small.

The dependence upon aggregation arises in that the ore may be coarse-grained or fine-grained, the former permitting closer sorting; in that it may be simple or complex, the latter calling for closer sorting where the grain allows; and in that it may occur as veins of high-grade mineral traversing worthless gangue on the one hand, or as a regularly impregnated mass on the other, the former permitting, the latter defying any separation by hand.

As already stated, when no mechanical dressing is done, hand separation may be continued upon ore as small as half an inch. Where, however, mechanical dressing is practised, the extent to which sorting is carried, apart from considerations of quality and aggregation, depends upon the cost of the operation in relation to that of mechanical dressing. In turn, the cost of sorting depends largely upon the lower limit of size to which it is carried; whereas it might cost 1s. 6d. per ton sorted out when this limit was $1\frac{1}{2}$—2 in., it would probably cost 4s. per ton sorted out from material between that size and 1 inch. There are, however, many plants where mechanical dressing costs less than this latter figure, and in such cases to pick material as small as one inch would be to exceed the economic limit of sorting.

It will be realized that into this question of the relative cost of the two operations the price of labour will largely enter; if that commodity were cheap and mechanical appliances costly, it would be rational to use hand separation to an extent otherwise not justifiable. In this same question the extent of the necessary plant is a considerable factor. To sort material having a minimum size of $1\frac{1}{2}$—2 in. does not make any large demand for plant, but where material finer than this is sorted, two stages to the operation are necessary, since fine material is not adequately displayed in the presence of coarse (Fig. 16). The additional plant thus rendered necessary would often cost more than an appropriate enlargement of the mechanical dressing plant, and to install it would be to carry sorting too far.

Finally, the extent to which sorting is carried depends also upon whether clean smelting ore or whether waste is being picked out. If it be the former, sorting must be credited with the fact that such ore will escape the losses inherent to mechanical dressing, and the operation can be carried farther on that account. If it be the latter, sorting must be debited with the fact that discarded waste always contains some mineral,

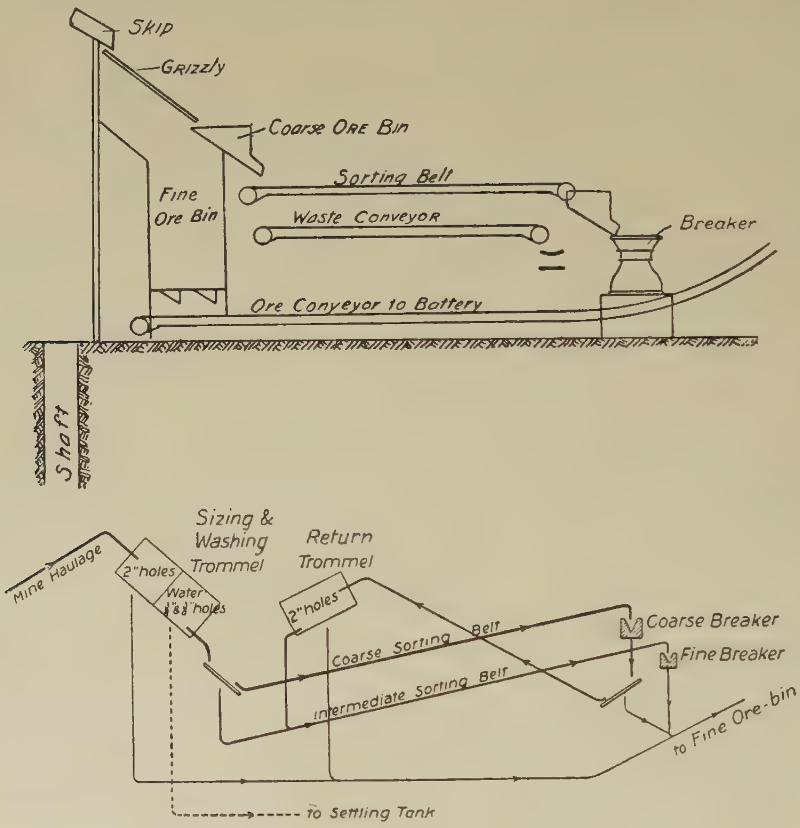


Fig. 16.

Sorting upon the Witwatersrand, South Africa.—Diagrams. The upper diagram illustrates the simplest practice. The ore is sorted only before and not after it has been broken. In addition to the sorting belt there is a special waste belt.

In the lower diagram stage breaking and stage sorting are illustrated, together with washing and sizing by trommels. Here again sorting is done on belts (pp. 32, 33).

and the extent to which it can be carried will be diminished by considerations of that loss.

In view of these many factors each case must be considered separately, and though sorting is general the practice is various. On the Witwatersrand gold-field an average of about 12 per cent of the ore sent to surface is discarded by sorting, the percentage at the richer mines being higher than that at the poorer mines, some of the latter not sorting at all; the cost per ton sorted out is about 1s. 4d., each native operator separating about 3 tons per day. At Tonopah, Nevada, treating a siliceous silver ore, the mines in the early days sorted some 20 per cent of shipping ore and 12 per cent of waste, the former at a cost of about 4s. per ton and the latter at about 2s. per ton, each operator separating about 5 tons per day. Similarly, at Cobalt, Ontario, where native silver is largely the valuable constituent, 2 per cent of rich ore was picked out containing half the silver. At the North Star gold mine on the Mother Lode, California, where the ore-body consists of auriferous quartz veins in slate, much of the latter being broken, 10 tons per man per shift has been sorted out, about 35 per cent of the crude ore being thus removed as waste. The Lake Superior copper mines, Michigan, generally discard but little waste by hand, this being also true for the large disseminated-copper deposits around Arizona. Other copper mines, however, discard as much as 15 per cent, besides separating an amount of smelting ore. At lead mines as a rule less sorting is done, because galena, being heavy and generally granular, is readily recovered by mechanical dressing. The same may be said of zinc, though on the Wisconsin field about 10 per cent of waste is discarded. Where, however, lead, zinc, and copper ores occur together, it is usual to find as much as possible of these ores separated by sorting. With tin ores very little sorting is done because, on the one hand, none could be picked out rich enough for direct smelting, and, on the other, it is generally most difficult to discriminate between ore and waste; exceptionally, at some of the rich mines in Bolivia different classes of ore may, however, be separated for different treatments. With respect to iron, at Lake Superior, at Cleveland, Yorkshire, at Bilbao, Spain, and elsewhere, the amount of material separated at the dressing floor as worthless may vary from 2 per cent to 25 per cent of the ore mined.

In conclusion, sorting, which requires only a relatively cheap installation, cheaply removes a substantial amount of material from the ore passing to the more costly mechanical-dressing plant, which may be smaller in consequence. Smaller size of that plant also follows from the fact, that in removing the extremes of rich ore and waste, the ore to be mechanically dressed is maintained more regular in value, a condition under which all the machines work best. Sorting, therefore, stands towards mechanical dressing much as dressing itself stands towards metallurgical treatment, and the arguments for and against the adoption of sorting are very similar to those for or against dressing in general.

CHAPTER III

COMMINUTION

Comminution is the whole operation of reducing the crude ore to the fineness necessary for mechanical separation, or for metallurgical treatment; for mechanical separation it effects the release of mineral from gangue; for metallurgical treatment it exposes the mineral so that attack by solution is possible, or it brings the ore to a size proper for fusion. Familiarly, Comminution is spoken of as Crushing, though that term may be reserved for the main and not the complete operation.

Complete Comminution may be divided into the following three stages:

Preliminary Comminution, or Breaking.
Primary Comminution, or Crushing.
Secondary Comminution, or Grinding.

Breaking

Breaking may be considered to be that preliminary stage of comminution which rarely sees any adequate release or exposure of the mineral (Figs. 44-50). It is effected by machines which apply pressure and work dry.

In certain initial stages of a mine's existence, or in places where labour is cheap and machinery dear, breaking may be done by sledging, spalling, and cobbing. By these means 1—3 tons may be broken per man per day at a cost of 1s. 6d.—4s. per ton. With a capacity so low and a cost so high, the possibilities of thus breaking by hand are strictly limited. In some respects, however, it possesses advantages over machine breaking, in that it does not make so large a proportion of fines, and the blow may be directed with intelligence. These advantages are particularly appreciated when hand-picking is the only separation employed, or when the resultant material is to be roasted in heaps. Where, however, the ore is hard or the quantity great, breaking must be done by machinery.

Breakers

Breakers are of three types, namely, Reciprocating or Jaw Breakers, Gyratory or Cone Breakers, Revolving or Disc Breakers.

Reciprocating or Jaw Breakers.—In these machines the ore is broken between a fixed and a moving jaw as the latter advances towards the former, while discharge takes place upon the succeeding retirement, when also fresh ore enters in readiness for the next advance (Fig. 17). By far the greater number of these machines are of the type known as the *Blake Crusher.* In the standard makes of these machines the reciprocating movement of the jaws takes place within a massive iron or steel frame of rectangular shape. At one end of this frame the fixed jaw is held vertically while the moving jaw hangs—from bearings situated on the frame—at an angle of 15°—20° toward the fixed jaw. In these relative positions the larger opening between the two jaws at the top is the "mouth," the smaller opening at the bottom is the "discharge," while the space between may be spoken of as the "throat."

The moving- or swing jaw, as it is termed, is maintained in this inclined position by a toggle-plate behind it, a piece so named because it takes part in the toggle action which is the feature of the machine (Fig. 19). The other end of this toggle-plate has its seat on an upright central member known as the pitman, which in turn is kept in position by another toggle-plate behind it, this second plate having its far seat fixed in the back of the frame. This pitman acts as a connecting rod between a heavy eccentric above and the toggle-plates below, raising the central toggle-seats till the plates approach the horizontal and then dropping them again, at each revolution of the eccentric shaft. As the pitman moves upwards the toggles spread at a constantly diminishing speed but with an ever-increasing force, outwards. Since also the far toggle-seat is held immovable in the framework the entire spread of the toggles is conveyed to the swing jaw. When the advance is complete and the pitman descends, the swing jaw is pulled back by a tension rod actuated by a stout spring or indiarubber buffer, its own weight helping. Discharge of the broken material then takes place, new material slipping in at the mouth. The eccentric shaft, having upon itself the driving pulley and being therefore the driving shaft, makes 150—250 r.p.m. in the larger machines, and 250—450 r.p.m. in the smaller, uniformity of effort during the revolution being maintained by two heavy fly-wheels, one on each side of the framework, which absorb energy during the retirement to give it out upon the advance. The actual speed at which a given crusher is run depends upon the hardness of the ore, a lower speed being effective with softer ore.

The dimensions of the two jaws and their disposition to one another, determine the size of material which the machine will take and the amount of *size-reduction* it will effect (Fig. 17)—the amount of size-reduction is expressed by the ratio between the size of the "feed" and that of the product. In particular, the length and width of the mouth determine the size of the feed, the former being the dimension from side to side of the machine, and the latter that from jaw to jaw. Ordinarily, the width is from 10 to 24 in. and the length 1·5—2 times the width, though both smaller and larger breakers exist, some indeed having a jaw-width of 60 inches.

With regard to the discharge, while the length of this opening is the same as that of the mouth, its width is but a fraction of the mouth width; moreover, the discharge width varies from a maximum upon the complete retirement of the swing jaw to a minimum upon its farthest advance, the amount of this movement varying from 0·5 in. in the smaller sizes to as much as 3 in. in the mammoth sizes. The maximum width is taken as the "set" of the machine, this set being usually 1½—3 in., but with exceptionally large machines as much as 6—9 in. The relation between the mouth width and the set determines the size-reduction, this being generally about 4.

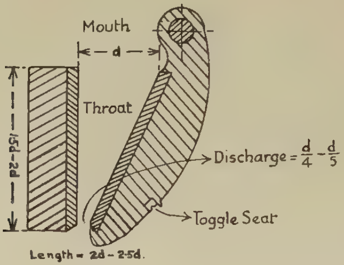


FIG. 17.

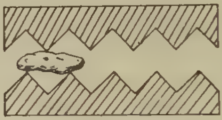


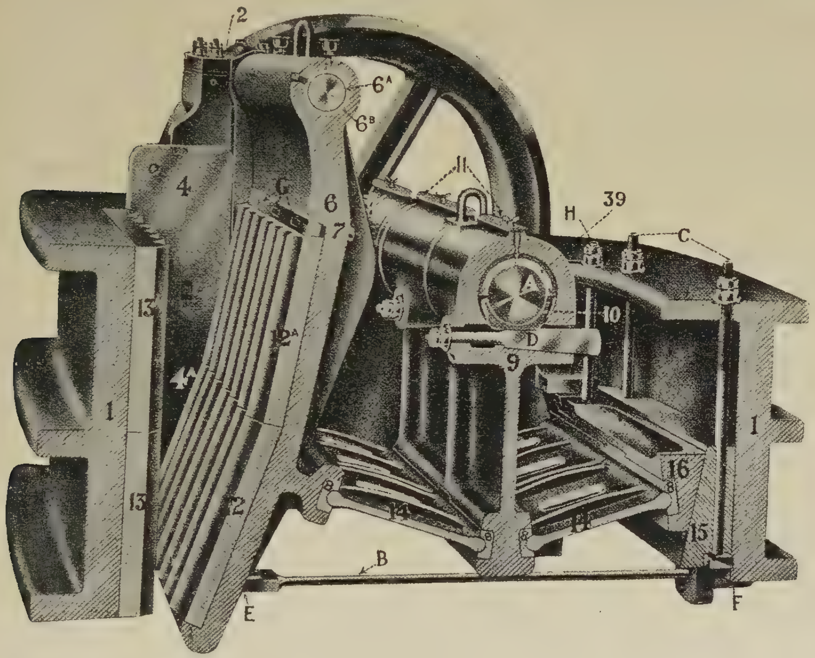
FIG. 18.

Jaw Breaker (Fig. 17).—Diagram illustrating the relative positions of the crushing faces. The face of the swing jaw produced upwards should pass under the axis of the shaft from which that jaw is hung, or at least not over that axis, for the movement of the swing jaw around an axis situated so low would have too large a component of lift and a smaller horizontal component. The harder and tougher the ore the greater should be the ratio of the depth of the crushing throat to its width (pp. 37, 38).

Jaw Breaker (Fig. 18).—Diagram illustrating the beam action made possible by corrugating the jaw plates (p. 41).

The depth of the crushing throat is an important factor, since with given dimensions of the mouth and discharge, the angle of the swing jaw from the vertical will depend upon this depth. At an ordinary angle, 15°—20°, and with the ordinary size-reduction effected, the depth is usually 2—2·5 times the width of the mouth. When the depth is less the swing jaw becomes more inclined, and there is a tendency for the material to be lifted instead of being seized; a shallow machine, though proper for weak material such as limestone, would not therefore be suitable for ore.

It is seen that breaking is done only upon the upward stroke of the pitman, corresponding to the advance of the swing jaw; accordingly, the machine is intermittent in its action, breaking only half the time. It is therefor that the fly-wheels are so essential. During the retirement of the jaw, energy is stored in these wheels to be of assistance when the heavy work of breaking has again to be met. Moreover, putting forth a doubled effort while working half time, the machine must be doubly strong ; in



A	Eccentric Shaft
B	Draw-back Rod
C	Adjusting Bolts
D	Wedge Cotters for Eccentric
E	Draw-back Rod Bolt
F	Draw-back Springs
G	Cotter Bolts for Swing Jaw
H	Bolts for Toggle Blocks
37	Bearing for Eccentric Shaft (Frame), not shown
39	Top Bearing for Eccentric Shaft (Pitman)
1	Main Frame (Cast Steel)
2	Cap for Swing Stock Shaft
3	Eccentric Shaft Bearing Cap
4	Cheek Plate (Top)
4A	Cheek Plate (Bottom)
6	Swing Stock
6A	Shaft for Swing Stock
6B	Bush for Swing Stock
7	Wedge for Swing Jaw
8, 8 8, 8	Toggle Cushions
9	Pitman (Cast Steel)
10	Bottom Bearing for Eccentric
11	Pitman Grease Covers
12	Lower Swing Jaw Face
12A	Upper Swing Jaw Face
13	Lower Fixed Jaw Face
13A	Upper Fixed Jaw Face
14	Toggles
15	Adjusting Wedge
16	Adjusting Toggle Block
17	Flywheel
18	Driving Pulley (not shown)

FIG. 19.

Cast-steel Jaw Breaker (Hadfield).—This illustration shows: frame reinforced by ribs ; swing jaw well extended upwards to keep the ore from flying into the mechanism, and jaw-plates steep at the top where the pieces are seized and given their first break ; jaw-plates corrugated, reversible, and sectionalized ; bearings for eccentric easily removable (p. 37).

addition to a firm foundation, the frame must be solidly cast or strongly braced, otherwise it will not withstand the immense stresses developed.

The most serious of these comes on the pitman, which, as it draws the toggles up to do their heavy work, is in tension—except where nipped by the toggles. The stresses it has to bear are, moreover, irregular, since it is rarely that the entire width of the jaw is at any one time equally engaged. To meet them it is made massive, generally of cast-iron in the smaller sizes and of cast steel in the larger sizes, but even with generous dimensions it occasionally fails. Uneven stress across the jaw has been minimized in one design by dividing the entire machine into two halves down the centre, so that within the same frame two narrow crushers exist side by side, working in unison; but this construction has not been endorsed by wide adoption.

The swing jaw more rarely fails since the tensile stresses to which it is subject are not so great as with the pitman; it is pushed up to its work. Moreover, since it neither hangs from a revolving shaft nor is raised and lowered at each revolution like the pitman, its weight may be greater without causing undue friction or irregularity of effort, and it may consequently be made most massive. Being inclined, it has a longer face than the fixed jaw, in addition to which, it extends upward to hang from a shaft usually placed high enough that the upper portion of the swing jaw becomes a bulwark against the flight of ore into the driving mechanism, and assists entry into the crushing throat. With the idea of securing a better entry, in one new type of machine the fixed jaw is inclined in the direction of the incoming feed, while the swing jaw hangs vertically.

Next after the pitman the eccentric shaft is liable to receive hurt when undue stresses are set up, as, for instance, when a hammer-head passes into the throat. When such happens, unless provision be made, either the pitman breaks or the eccentric shaft becomes bent. To escape these costly injuries some cheaper breakage is usually provided. For instance, in some designs the driving pulley is not keyed to the eccentric shaft but bolted to a collar or to a fly-wheel which is so keyed, this weaker connection becoming sheared by excessive stress. Or, the toggle-plate may be constructed of two overlapping halves riveted together in such a manner that an undue stress shears the rivets.

Speaking of the machine as a whole, the jaw breaker is simple, reliable, and requires little height. On the other hand, it is heavy, and a firm well-braced framework is necessary for its support, or a destructive vibration will develop. Large crushers of this type are, therefore, set on solid foundations, rock or concrete.

Apart from breakage there is wear of the breaking faces. This wear is taken on special jaw-plates wedged or bolted into position on the actual jaws and easily removable. To prevent these plates from breaking, their seat upon the jaw is made perfect, either by running in soft metal behind them or by planing the faces. Ordinarily the jaw-plates present a plane surface to the ore, but in the larger sizes a corrugated surface is often used, the solid angles of such corrugations being usually 90°, and the pitch from 2 to 6 in., or more, the larger figures applying only to very large breakers (Fig. 18). With such corrugation, a large piece spanned across any two ridges of one jaw would be readily broken at its middle by the advance of an opposing ridge. Beam action and consequent tensile-stress will effect quite readily what pure compression would find great difficulty in doing; rocks are weak to tensile stress and strong against compression.

Most wear taking place at the bottom these plates are usually so shaped that they can be reversed. In the larger sizes this endeavour to minimize the weight of metal discarded in worn plates is carried further by making these plates sectional, the sections interlocking when in position but easily removable (Fig. 20). With the same idea large plates are ribbed at the back instead of being solid right through.

Just as the jaw-plates protect the jaws, so the sides of the throat are lined with what are known as cheek-plates which preserve the main frame, these being shaped to suit the dimensions of the throat; extending above, they also preserve the top of the frame and incidentally prevent ore from leaping the side.

To set the width of the discharge to any desired figure, and at the same time to keep this set constant through all conditions of plate wear, appropriate adjustments are provided. To this intent the far seat of the back toggle is laid in a wedge-shaped block capable of being moved forward or backward, by the up and down movement of a wedge behind it; as this toggle seat is moved forward or backward, so also is the swing jaw. In addition, the toggle-plates may be changed for others which are longer or shorter, this change constituting a rougher adjustment.

Concerning the material of which these machines are constructed, it is usual to find the frame of cast-iron when the machines are small, and of cast steel when they are large. Cast steel, with suitable strengthening ribs or bolts, permits a reduction in weight by about one-half (Figs. 19, 20). The swing jaw, pitman, etc., being similarly of cast iron or steel, the whole machine, including the two fly-wheels, is very heavy, a size with a mouth $10'' \times 20''$ weighing about 8 tons, and one of $24'' \times 36''$, when of steel, weighing about 40 tons. With a cast-iron frame the weight of a breaker expressed in tons is about the same figure as the capacity of the machine expressed in tons broken per hour.

The material of the jaw-plates must be chosen to suit the ore. With a material so brittle as limestone chilled cast-iron would have a sufficiently long life, and being cheap would probably be preferred. With hard ore manganese steel, costing almost four times as much, would probably be more economical, while chrome steel and forged steel would suit intermediate conditions. In the form of jaw-plates manganese steel would cost about 6d. per lb., forged steel 4d., cast steel 2½d., and chilled cast-iron about 1½d. per pound. The ordinary amount of wear, including the weight of metal in the discarded plate, is about 0·1—0·2 lb. of average steel per ton broken. Of manganese steel, which if properly heat-treated is extraordinarily tough rather than very hard, the consumption by wear would be substantially less.

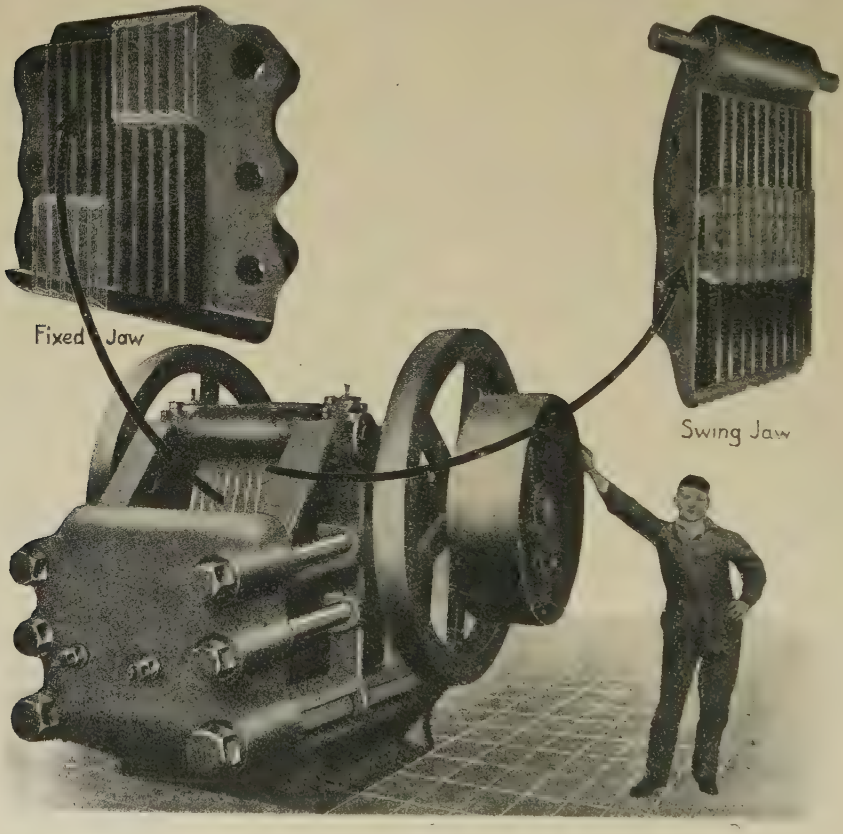


FIG. 20.

Sectionalized Jaw Breaker (Joshua Hendy)—Frame held together by strong bolts; interlocking and interchangeable jaw-plates (p. 41).

The capacities of these breakers vary from 5 tons per hour for a small size, to as much as 500 tons per hour for the mammoth sizes; for common sizes it is generally about 0·1 ton per square inch of mouth area. In relation to power, the capacity varies from 0·5 ton per hour per horse-power installed, in the smaller sizes, to 2 tons in the larger sizes, with an average of about one ton. Breaking finer than 1½ in. the capacity of a machine drops considerably.

Concerning the quality of the work done, the product of these breakers is not particularly regular, partly because the discharge opening varies in width from a maximum to a minimum, but also because long flat pieces may pass. It is sometimes stated that a more regular product is obtained by passing all the ore, fines and lumps together, through the breaker, keeping thereby the throat so filled that flat pieces have not the freedom to align themselves with the length. In general, about 15 per cent of the product is larger than the set of the machine.

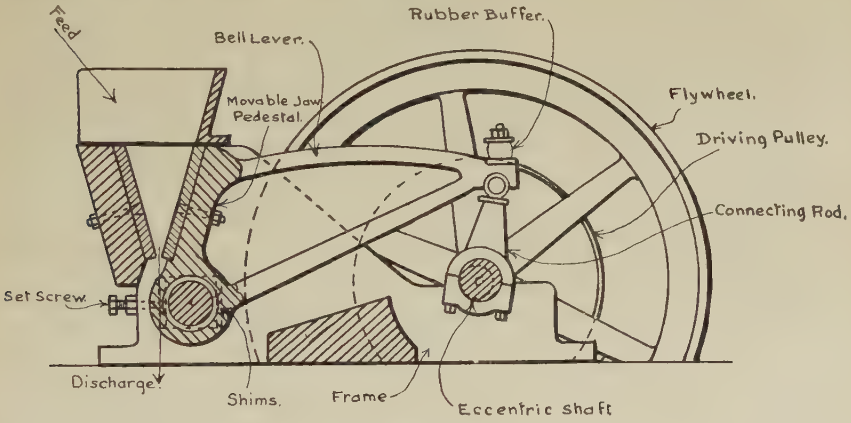


Fig. 21.

Jaw Breaker, Dodge Type.—Sectional view (p. 43).

The cost of breaking by these jaw breakers is generally about 2d. to 4d. per ton broken, the lower cost being associated with large tonnage. With very small plants the cost will be greater still; with very large plants still less.

Apart from jaw-plate renewals, lubricating oil and bearing metal are the principal supplies needed. These are consumed largely to relieve the friction of the eccentric shaft and the toggle bearings, adequate lubrication of these members being most necessary.

A second type of jaw breaker, the *Dodge Crusher*, is in many respects similar to the Blake; it has the same principal parts all contained within the same massive rectangular frame (Fig. 21). It differs, however, from the Blake in that the moving jaw is not hung from above but supported from below, while the connecting rod, instead of being a central vertical member depending from the eccentric shaft, is at the far end of the machine and sits on that shaft, its motion being conveyed to the jaw by a massive bell-lever of which the jaw is actually the short arm; moreover, the toggle-plates are absent. The eccentric shaft is again the driving shaft, and again there are two heavy fly-wheels.

The principal difference caused by the above reversed disposition of the principal parts is that the movement of the swing jaw is greatest, say $\frac{3}{4}$ in., at the mouth, and least, say $\frac{1}{4}$ in., at the discharge; with the Blake it was contrariwise. This difference has distinct consequences. Since the discharge opening remains practically constant, the machine delivers a more regular product; but, lacking movement at discharge, it more readily chokes and its capacity is less. Again, the largest pieces being broken farthest from the fulcrum and the bell-lever transmitting the force being long, this lever requires to be extraordinarily massive, and the running of the machine is very unbalanced. Accordingly, the Dodge Breaker is not made so large as the Blake. As an intermediate breaker to obtain a regular product it has some application, but compared with the Blake it is an unimportant machine.

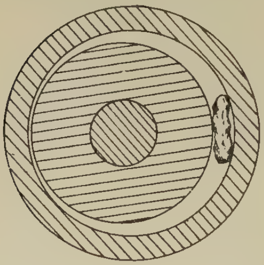
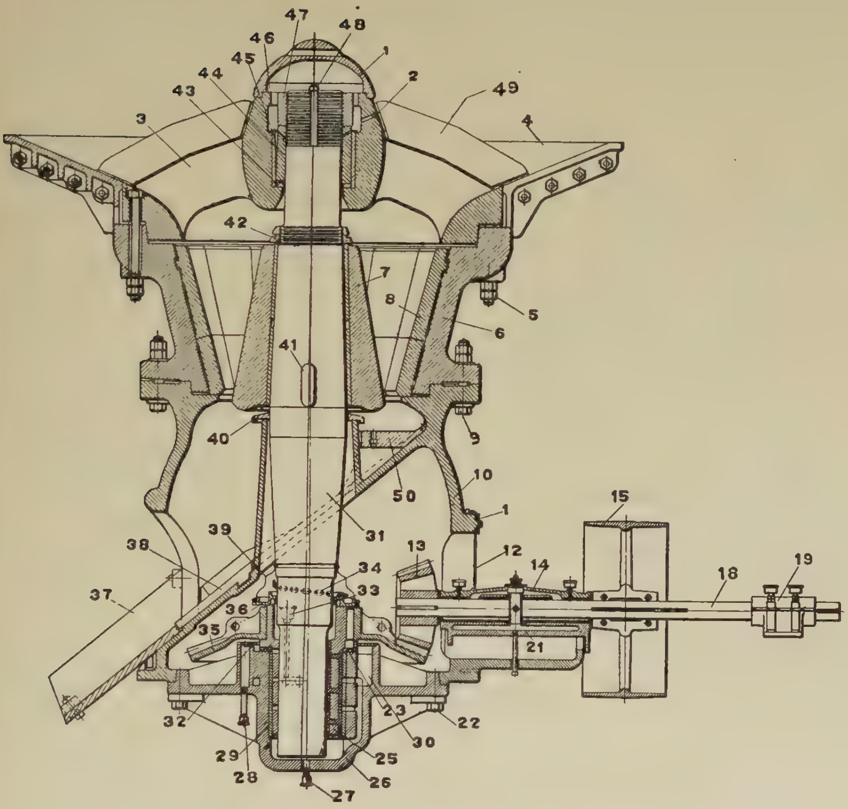


FIG. 22.

Gyratory Breaker. — Diagram illustrating relative position of crushing faces and the possibility of applying beam action. The outer ring represents the concave plates and the fixed jaw; the inner ring represents the conical head and the moving jaw; the central circle is the gyrating spindle (pp. 44, 46).

Gyratory or Cone Breakers.—In breakers of this type the ore is caught and broken inside a ring of concave jaw-plates by the gyration of a conical head within that ring (Fig. 22). The concaves represent the fixed jaw of the Blake, and the conical head the moving jaw. Usually the interior concave surface is not vertical but inclines inward like the surface of an inverted cone (Fig. 23).

In the Gates Crusher and similar machines—Gates being the name under which these machines became prominent—the concaves are held in a cast-steel shell which constitutes the central portion of the machine. Upon this central shell comes an upper shell which, expanding at the top, forms a flat hopper to receive the ore and direct it to the centre. Across this hopper the machine is spanned by two or three massive arched arms, which meet at the centre to form a hub wherein the upper end of the spindle carrying the conical head is held. The mouth of the machine is the upper annular opening between the concaves and the conical head. The



List of Parts

1	Spider Cap
2	Oil Canal
3	Spider
4	Hopper
5	Spider Bolt
6	Top Shell
7	Head
8	Concaves
9	Middle Joint Bolt
10	Lower Shell
11	Door Pin
12	Door
13	Bevel Pinion
14	Cap for Countershaft Bearing
15	Band Wheel
18	Countershaft
19	Outboard Bearing
21	Countershaft Bearing
22	Bolt for Bottom Plate
23	Brass Wearing Ring
25	Eccentric
26	Bottom Plate
27	Drain Pipe
28	Overflow Pipe
29	Steel Bushing
30	Oil Chambers
31	Main Shaft
32	Dust Plate
33	Oil Cup
34	Oiling Collar Chain
35	Bevel Wheel
36	Oiling Collar
37	Discharge Spout
38	Wearing Plates
39	Canvas Hood
40	Dust Collar
41	Feather Key in Main Shaft
42	Lock Nuts on Head
43	Wearing Ring
44	Sleeve
45	Steel Bushing
46	Annular Oil Ring
47	Adjusting Nut
48	Key
49	Shield for Spider
50	Diaphragm Liner

Fig. 23.

Gyratory Breaker (McCully). — Vertical Section. This illustration shows the spindle hung from above by a nut or collar caught in the central hub (p. 44).

width of this mouth is that of this annular opening; its length is the outside circumference, but as this is divided by the above-mentioned arms, it is usual to give the length of the segment between any two arms, and the number of such segments, whether two or three. The width largely determines the size of feed the breaker will accommodate; the long length gives great capacity.

Below the central shell comes a third section, which increases somewhat in diameter downwards, its bottom rim being bolted to the foundations. On one side of this lower shell the broken material emerges; on the other the horizontal driving-shaft enters; while within is contained the lower end of the spindle, the eccentric by which it is gyrated, and the driving gearing, all these parts being boxed-in by a bottom plate or basal casting.

The driving shaft, running at a speed of 350—500 r.p.m., has on its internal end a bevel pinion, which drives a horizontal bevel-wheel at a speed reduction of about 2·5 : 1. This wheel drives a shaft or journal of large diameter borne in a bearing in the basal casting. Inside this journal—generally described as the eccentric—the spindle of the machine is held eccentrically so that it gyrates at the same speed as the wheel rotates, namely, 140—200 per minute, the radius of gyration being 1—3 in.

The conical head, moving with the spindle, gyrates within the concaves, approaching and receding from each section of the entire ring in turn. The spindle being held at the top, the greatest movement within the crushing throat is, as with the Blake, at the bottom, this movement being usually $\frac{1}{2}$—1 in. From jaw breakers, however, this type differs essentially in that, whereas in the former breaking only takes place half the time yet over the complete surface, with gyratory breakers it takes place all the time yet over only a portion of the complete breaking surface at any one time. The distinct consequences of this difference are, that the work of breaking is done more uniformly and, for a given amount broken, the maximum stress is less; accordingly a machine of this type is less heavy than one of the reciprocating type having the same capacity. In this gyratory progress around the circle there is also a rolling motion which tends to prevent packing of the material, while, in addition, the convex head is able to apply beam action across the concavity (Fig. 22). These crushers consequently employ power more efficiently, are generally less troubled by packing and choking, and deliver a more regular product; while for the reason given above, and for the additional reason that no fly-wheels are required, the same weight of machine possesses considerably greater capacity.

Gyratory breakers are essentially large-capacity machines, no standard machines of small capacity being made. This attribute follows from the relatively great length of mouth, a particular width of mouth being associated with a much greater length in the gyratory than in the reciprocating breaker; moreover, the width cannot be increased without at the same time increasing the length, the two being interdependent. For its full working the gyratory requires to be fed all round the circle, or centrally on to the spindle-hub.

Adjustment of the set of this machine is made by raising the spindle, either by screwing it up from above or by forcing it up from beneath. This means, however, does not allow more than about ½ in. alteration of the discharge width, and greater adjustments must be made by putting in a new set of concaves. Some operators start a new head low down within old concaves, and adjust for wear first by raising the head and then by changing the concaves. Unless some procedure of this sort is followed, only the lower portion of the head gets worn, and since these heads are not reversible, a discarded head contains a large proportion of its original weight. To minimise this loss the head is generally made of two portions, an inside core of ordinary metal and an outside shell or mantle of the steel found best suited to the particular conditions. Sometimes the head is made of an upper and a lower section, but this is seen more often with the concaves. Since beam action is obtained by the concavity of the throat, corrugated surfaces are not so necessary with these machines as with jaw breakers, nor are they much used. One hundred thousand tons is quite a reasonable figure to expect the head of a moderate-sized machine to crush before being discarded.

It will be realised that the spindle and the manner in which it is held are important elements in this type of breaker. In the early types this spindle was supported on a hard-steel button in the bottom plate, which button by a screw could be raised or lowered, and the spindle with it. To-day, the spindle is more usually suspended from a collar screwed on to its upper end, and bearing upon a shoulder in the hub, the spindle being raised or lowered by screwing this collar. In either case, in order to avoid such angularity as would cause difficulties in the hub-bearing above and the eccentric-bearing below, the spindle is kept as long as other factors permit. It is in consequence a long member, and being long it must be immensely strong to stand stiff under the tremendous stresses to which it is subjected. To meet this necessity it is of greater diameter in its central portion than at either end, and on this enlarged portion, which is generally polygonal in section, the head is seated.

Concerning the materials of construction, the body shells are usually of cast steel, while the jaw-plates are of manganese steel, chrome steel, forged steel, cast steel, or chilled cast-iron, as the character of the ore and the respective costs of these materials determine. The wear per ton is ordinarily less than with jaw breakers—probably because of their greater size and capacity—but the removal and replacement of the plates is much more difficult, so that the total repair cost is somewhat higher. Should the spindle break, which sometimes happens, the cost of its removal and replacement adds heavily to the operating cost; in addition, the withdrawal of the spindle requires considerable height beyond the relatively great height of the machine itself (Fig. 24). This is one of the disadvantages of the machine. Against it, however, may be set the fact that the effort being continuous and the movement circular, no great stress or vibration comes upon the foundation, which consequently may be less massive.

In regard to operating cost, while that of repair and renewals is perhaps heavier, that of power is substantially less than with jaw breakers. The quantity crushed per horsepower-hour generally varies from 0·6 ton for the small sizes to 4 tons for the largest sizes, equivalent to, say, 1·2 tons for the medium machines more often found on mines. As to attendant labour, these machines being larger units require appreciably less attention per ton than jaw breakers, in addition to which, as before mentioned, the wide basin-shaped top acts as a convenient chute for the entry of the material. With jaw breakers considerably more attention in feeding is required. Altogether the cost of gyratory breaking may be said to be from 1d. to 4d. per ton crushed.

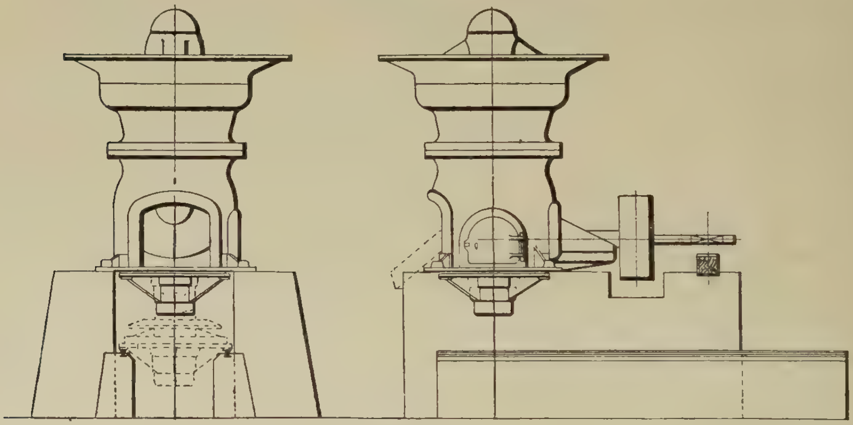


Fig. 24.

Gyratory Breaker.—General Outline. This illustration indicates in side and front elevations how the bottom plate may be dropped and moved to one side on rails when it is desired to lower the spindle (p. 48).

Of late years a gyratory breaker of a different construction and known as the Telsmith Breaker has come into use (Fig. 25). The spindle of this breaker is short and thick; moreover, it neither gyrates nor rotates, but is at both ends firmly borne in the main casting of the machine. Upon it a long eccentric sleeve, by gearing at the bottom, is made to revolve. On that eccentric sleeve the head is slipped and, upon revolution, gyrated. The motion of this head is therefore a parallel one, neither greater nor less at top or bottom; the spindle exercises no lever action. In the ordinary gyratories the spindle is long, the power applied at one end, and the work done nearer the other, conditions which tend to spindle breakage.

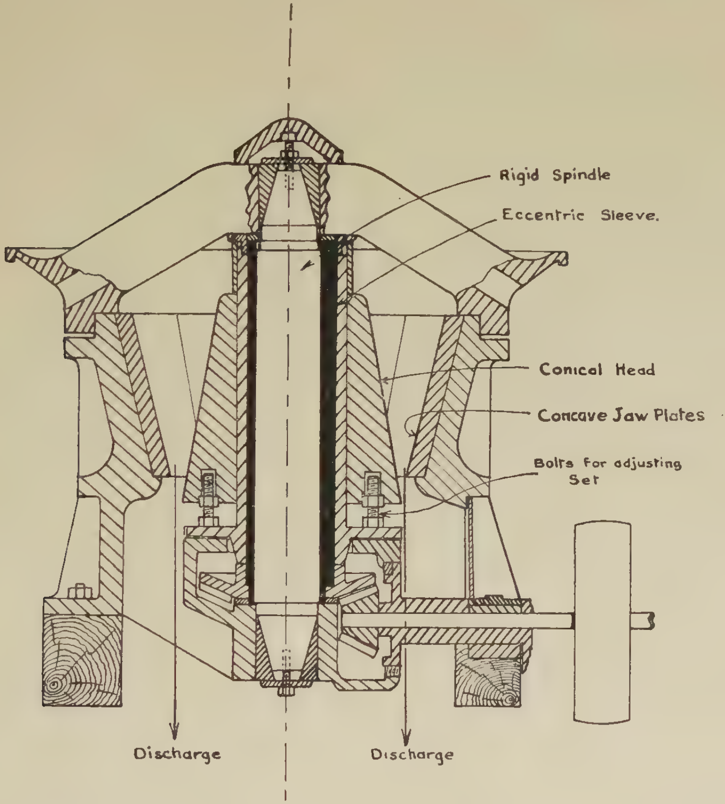


Fig. 25.

Short-spindle Gyratory Breaker (Telsmith).—The eccentric sleeve which causes the gyration of the head stands out clearly; this being a parallel movement, no question of angularity of spindle arises (p. 48).

Another advantage of this machine is that it requires considerably less height, the machine being shorter in itself and requiring less head room for convenience in repairs. Since also there is no element of lift in the movement, it is stated to bite the ore quicker than the ordinary type, and seizure is more certain.

Concerning the relative spheres of reciprocating and gyratory breakers, it may be said that where few extremely large pieces come with the general run of ore, it is better to employ jaw breakers for these, because a gyratory with the necessary width of mouth would possess a capacity far beyond that necessary (Fig. 49). On the other hand, gyratory breakers are well suited to take the lumps ordinarily resulting from mining, except where the quantities are small, when jaw breakers would be better. For fine breaking, gyratory breakers of standard design are not suitable and a special short-headed machine must be used (Fig. 49), or preferably a jaw breaker; small material does not offer the same possibility of applying beam action, and the gyratory breaker then loses that advantage. Where the ore is flaky or breaks into slabs, a jaw breaker would be liable to give too irregular a product.

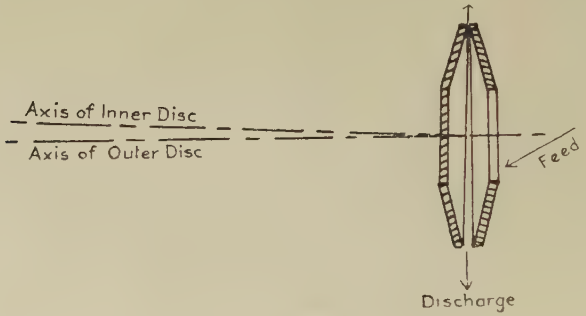


FIG. 26.

Disc-breaker.—Diagram of crushing jaws (pp. 50, 51).

Rotary- or Disc-breaker.—With the disc-breaker the ore is fed centrally between two rapidly-revolving saucer-shaped discs, placed rim to rim, and enclosing between them a space relatively wide at the centre and narrow at the rim (Fig. 26). This narrow width at the rim is not of constant dimension around the circumference, but, one disc lying and rotating in a slightly different plane from the other, varies from a maximum, which is the set of the machine, to a minimum at a point diametrically opposite. The ore fed through the centre of the outer disc is thrown centrifugally outwards to the rim, where such pieces as are too large to pass out are caught, to be carried round and broken as they arrive in the zone where the two discs approach.

This type of breaker possesses two important advantages, firstly, that the throat or breaking space, decreasing in width yet increasing in diameter, is of practically constant volume from intake to discharge, whereas in the reciprocating and gyratory breakers this space regularly decreases as the material sinks deeper into the throat, a condition engendering choking and limiting the possible size-reduction; and, secondly, that the centrifugal force developed, effects a positive and immediate discharge of such pieces as are fine enough, leaving thereby the throat free for the better breaking of those which remain.

Symons Disc-breaker.—In the horizontal type of the Symons disc-breaker the two discs, an outer and an inner, are held in place at the ends of two shafts, one of which is hollow and contains the other in such a manner that the two axes are at an angle (Fig. 26). In addition, the far end of the contained shaft can by an eccentric be made to gyrate within the hollow shaft, the plane of the inner disc at its other end being thereby constantly altered. This gyration is made possible by a ball-and-socket bearing, the ball being on the contained shaft and the socket on the hollow shaft. Finally, the shaft and the outer disc bolted to it can be rotated by means of a pulley around the former (Fig. 27).

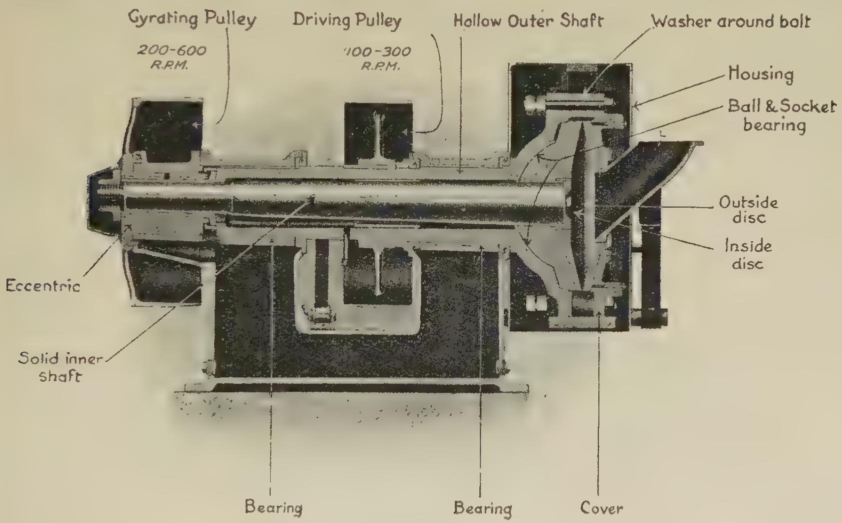


FIG. 27.

Horizontal Disc-breaker.—Longitudinal section (Hadfield) (p. 51).

In greater detail the outer disc is held in a cap firmly bolted across to the socket of the hollow shaft, while the inner disc is fixed to the ball of the contained shaft. The latter shaft has no separate rotation, but is driven through grip with the ore, this grip being sufficient to cause the two shafts to rotate at the same speed, a speed which varies from 100 r.p.m. in the larger sizes to 300 r.p.m. in the smaller sizes. If the axis of the contained shaft were fixed, then in one complete revolution there would be but one approach of the two discs, that is to say, but one crushing effort; by gyrating the far end the number of these efforts is multiplied, and the effort instead of being made always at the same place is made in succession all round the circle, giving a multiplied duty and even wear. The eccentric which effects this gyration is therefore driven 200—600 r.p.m., in the opposite direction to the main rotation.

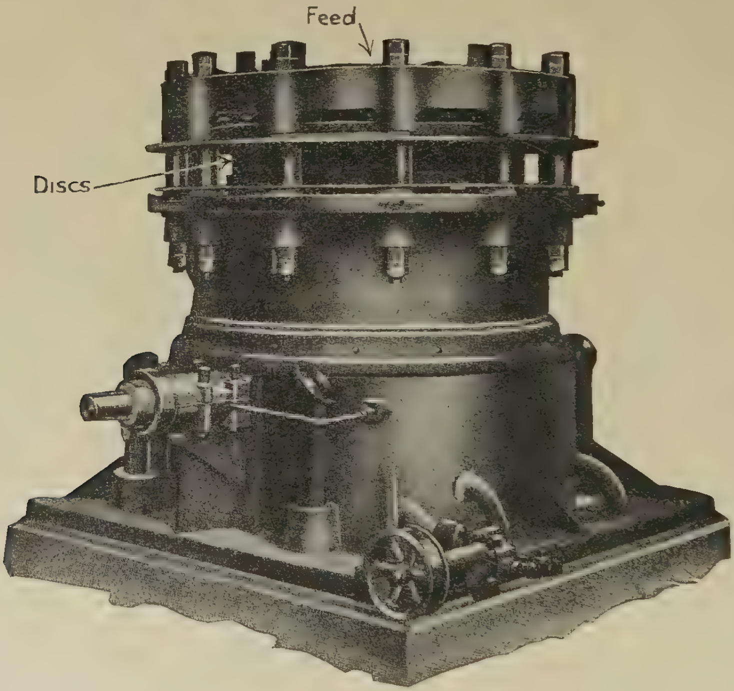


Fig. 28.

Vertical Disc-breaker.—General Appearance. In this illustration the housing and discharge chute have been removed. The prominent vertical bolts are those which hold the outer cover and its disc close up to the inner disc. The rotation around the central vertical axis is effected by bevel gearing within the machine. The driving shaft is shown with the driving pulley removed. These vertical breakers are used to break somewhat finer material than those of the horizontal type (p. 53).

The adjustment of the crushing discs is made by changing the shims which act as distance-pieces to the cap holding the outside disc. The crushed material flies out from between the discs into an encircling chute by which it is conducted to the discharge, provision against the escape of dust being made by a suitable housing. The discs themselves are generally of manganese steel. Care must be taken to keep foreign material, steel for example, from getting between them.

These disc-breakers are made in sizes varying from 10 in. disc diameter to 48 in. diameter, the larger machines being those most favoured. Corresponding to such a range in diameter the space between the discs at the centre varies from 2¼ in. to 8 in., while the set varies from ⅜ in. to 1 in. Upon material already broken to 2—3 in., the capacities vary from 5 to 25 tons per hour, and the power consumption from 10 to 50 h.p. equivalent to 1000 lb. per h.p. per hour. From these figures it is seen that these machines are fine breakers. They often follow ordinary breakers to carry the work a stage further (Figs. 47, 50).

In the horizontal design this machine lies upon a hollow base serving as an oil-container, whence oil is forced by pump through all the principal bearings to accomplish the important work of lubrication. Later machines made particularly for breaking material of 1—2 in. to, say, $\frac{1}{16}$ in. at a rate of about 15 tons per hour in the largest size, have been made vertical (Fig. 28). Whatever the design, the inclusive weight of a machine in tons is roughly equal to the number of tons the machine is capable of crushing per hour. The over-all dimensions of this machine are small for the work it does.

PRIMARY CRUSHING AND SECONDARY GRINDING

Crushing, as the primary stage in comminution, connotes such a substantial release or exposure of the mineral that mechanical separation or metallurgical extraction may proceed; it will be remembered that breaking rarely reached that position. Similarly, *grinding* or secondary comminution, when used, completes the release or exposure of the fine mineral, often after a first separation has been made. As these two stages overlap their separate description is neither convenient nor demanded.

Comminution may be effected either by pressure, typically employed with breakers; or by shearing, as with grinding machines; or by impact, as with the falling stamp. Pressure having been the particular force-application in the breaking machines so far described, to maintain continuity those crushing machines which use pressure are first described.

Rolls

A set of rolls consists of two similar discs or rollers revolving in the same plane but in opposite directions, upon horizontal shafts so placed that the two opposing roller-faces are parallel and within a “ set ” distance of one another. Into the space above this set the ore is fed, by friction seized and by pressure broken, and then, when in course of revolution the plane containing the axes of the two rollers is reached, discharged, the crushed product falling clear (Fig. 29).

Crushing in this way rolls make use of toggle action. Revolution at a regular speed around the circle connotes a decreasing speed of approach of the two points, one on each roller, upon which a piece of ore between the rolls would be supported, and by which, under proper adjustment, it would be seized. The energy available for crushing is that which would maintain uniform revolution, part being stored in the rotating masses, part being directly received from the prime mover. Around the circle this energy may be taken to be constant. That being so, and the movement inwards of the two points mentioned above decreasing with descent to the axial plane, the force available for crushing continually increases to that plane. A piece of ore once seized will accordingly be broken, or, if too strong to surrender, will force the rollers apart and pass.

Seizure accordingly is all-important. It depends upon one natural factor, the coefficient of friction between the ore and the material of the roller, and upon several geometrical factors residing in the dimensions and disposition of the rollers. Seizure results when ore and rollers do not slide in respect to one another. Considering only one side of the position, since what is true of one is true also of the other, the roller-face at the point of contact is moving along the plane tangent at that contact, and the angle between that plane and the vertical plane midway between the two rollers may be regarded as that of the classic incline-plane of mechanics, where the relation between the coefficient of friction μ and the sliding angle a is expressed by the formula $\mu = \tan a$. So long as the angle between the two planes mentioned is less than satisfies this relation, there will be no sliding, the piece will be seized (Fig. 29).

Or, considering now the two symmetrical halves of the position, so long as the "wedge angle" subtended below the piece of ore and between the two tangents, is less than twice the sliding angle, seizure will take place. This critical angle, equal to twice the sliding angle, may be designated the "angle of nip." In order that the ore shall be seized, the wedge angle must not be greater than the angle of nip.

This wedge angle depends upon geometrical factors: the size of the ore, the diameter of the rollers, and the set distance between them. For the same diameter and same set it varies directly with the size, and a piece above a certain size will not be seized unless one at least of the other factors is suitably altered; for the same diameter and same size of material it varies inversely as the set; finally, for the same size of ore and the same set it varies inversely as the roller diameter.

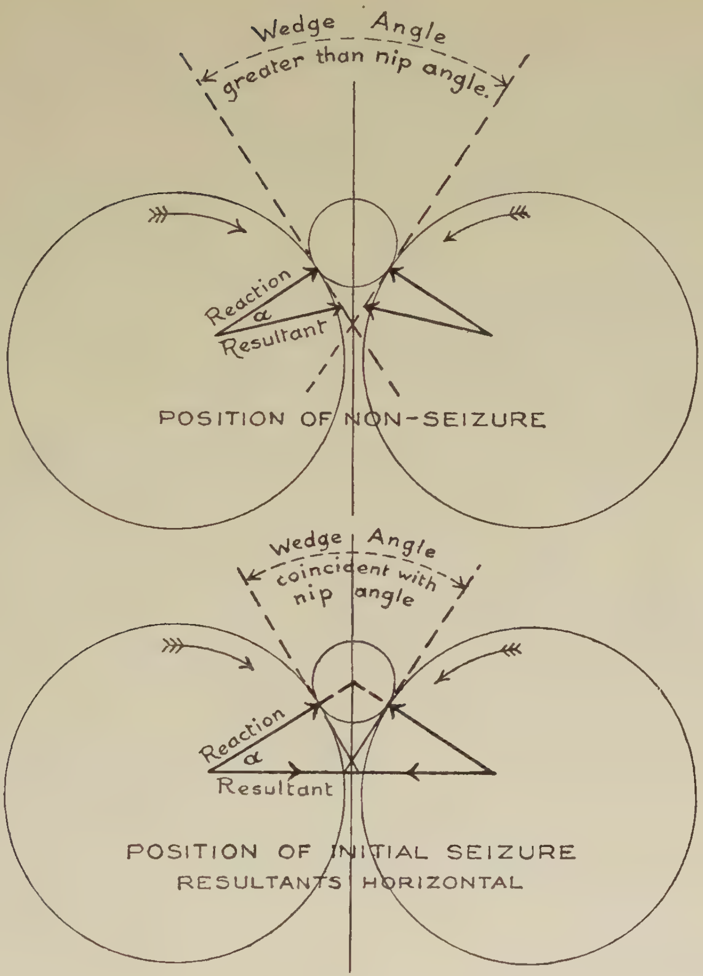


FIG. 29.

Rolls.—Diagrams illustrating the relative positions of the crushing faces, and the forces which determine seizure. The upper diagram shows a position of non-seizure; the lower illustrates the position when seizure first becomes possible (p. 54).

Consider a piece of ore supported between two inwardly-revolving rollers: half the weight is supported on each, not directly from beneath but by a reaction along the radius joining the roller axis to the point of contact. Friction, which is the other force acting on the piece—and in amount is the product of the reaction and μ the coefficient of friction—tends to carry the piece down. Reaction and friction are at right angles, and the hypotenuse which joins them represents their resultant. Further, since by definition $\mu = \tan \alpha$, where α is the sliding angle of mechanics, and $\dfrac{\textit{Friction}}{\textit{Reaction}}$

is also equal to μ, the angle a between the reaction and the above-mentioned resultant must be the sliding angle.

This resultant may be inclined upwards, downwards, or be horizontal. If inclined upwards the two resultants, one on each side, would tend to lift the piece, whereas if inclined downwards there would be seizure, the limiting position being the horizontal. But in the horizontal position it is clear from the diagram that a is equal to half the 'wedge angle' under the particular circumstances; in other words, when seizure takes place, the particular wedge angle, then known as the 'angle of nip,' is equal to twice the sliding angle a.

The same result would have been reached if the greater reaction arising from the crushing strength of the piece of ore had been considered. Accordingly, seizure once effected by friction due to weight, becomes fracture as the energy within the rotating masses comes into play.

For convenience, in the lower diagram a higher coefficient of friction has been introduced; such, for instance, as could be obtained by roughing the roller surface. The angle of nip varies with this coefficient.

All these factors were brought by Weisbach into the expression

$$D = \frac{1}{1 - \cos a}(d - a),$$

where D is the minimum permissible roller-diameter, d the size of the ore pieces, a the set of the machine, and a the sliding angle corresponding to the particular coefficient of friction. This formula gives the roller-diameter suitable for large material. Small material will be satisfactorily accommodated by any diameter convenient for the large.

Seeing that difficulties of construction increase rapidly with roller-diameter, it is necessary to keep this dimension low. Yet, while maintaining a set as small as possible, rolls should take ore as large as possible, these two factors determining the size-reduction; the factor $(d - a)$ to which diameter is proportional, cannot accordingly be unduly reduced. It remains, therefore, to keep the other factor, $\frac{1}{1 - \cos a}$, as low as possible, which can be done by selecting such conditions and such roller material that the coefficient of friction between roller and ore is high.

This coefficient of friction is influenced by several factors. The material of the roller should not be too hard or it would become polished with wear and the friction would be reduced; in this respect, hard steel and hard cast-iron give place to mild steel. Again, a wet roller has a bite which a dry surface does not possess, as witness the necessity of moistening the stone to sharpen the knife. A friable ore quickly provides the sand necessary to increase the friction, whereas a hard siliceous material would not so readily give of itself for that purpose. Finally, the speeds of the roller and of the falling ore influence the position; when the roller speed is high, part of the friction is lost in overcoming the inertia of the ore, the effect being equivalent to a reduction in the coefficient; on the other hand, if the ore were dropped from a height, it would of itself press deeper between the rollers, increasing the friction. Ordinarily the coefficient of friction is about 0·33, at which figure the sliding angle would be 18½°, and the angle of nip 37°.

Practice confirms the above considerations; it is usual to find the wedge angle to lie between 25° and 40°, when calculated with respect to the maximum size of feed. Using an angle of 37° the formula given above may be simplified to

$$D = 20(d - a).$$

From this formula it will be seen that with increasing size of feed the necessary roller-diameter increases more rapidly. Here also experience agrees; it is found better to reduce material larger than about 2½ in. by means of breakers than by rolls. It can be further seen that with the same size of rolls a greater ratio of size-reduction is possible with smaller than with larger material, the set being suitably altered.

ROLL TYPES

Geared or Cornish Rolls (Fig. 30).—In this type the two rollers of equal diameter, generally 15—36 in., are 9—18 in. wide, one roller being sometimes a little wider than the other. One of them, the wider if such there be, revolves in fixed bearings situated on a rectangular frame, being directly driven by gearing, while the other revolves in movable bearings being driven by gearing from the first. The two rollers are set about ½ in., or less, apart, the movable roller being kept from coming closer, generally by distance-pieces, known as shims, placed between the two pedestals, and from separating farther by powerful springs which only yield when some unyielding mass, such as a piece of steel or an excessive amount of material, would pass.

These rolls are run at a speed of 15—25 r.p.m. equivalent to a peripheral speed of 100—250 ft. They have a capacity of 4—8 tons per hour reducing 1½—2 in. material to a product one-quarter that size. Requiring 10—20 h.p. they crush about 800 lb. per h.p. hour, to pass a screen the oversize from which is returned to be crushed again. When the finished product is thus determined by a screen, the rolls and screen are said to form a "closed circuit" (Fig. 43). Working in "open circuit," that is, with but a single passage through the rolls, the capacity would be appreciably greater; the product, however, would be less regular.

Though this type is open to the objection that the toothed wheels driving the movable roller cannot be efficiently engaged throughout all the shifting necessary to maintain constancy of set as the rollers wear, this type is still in considerable use, notably at Broken Hill (Fig. 48). It is

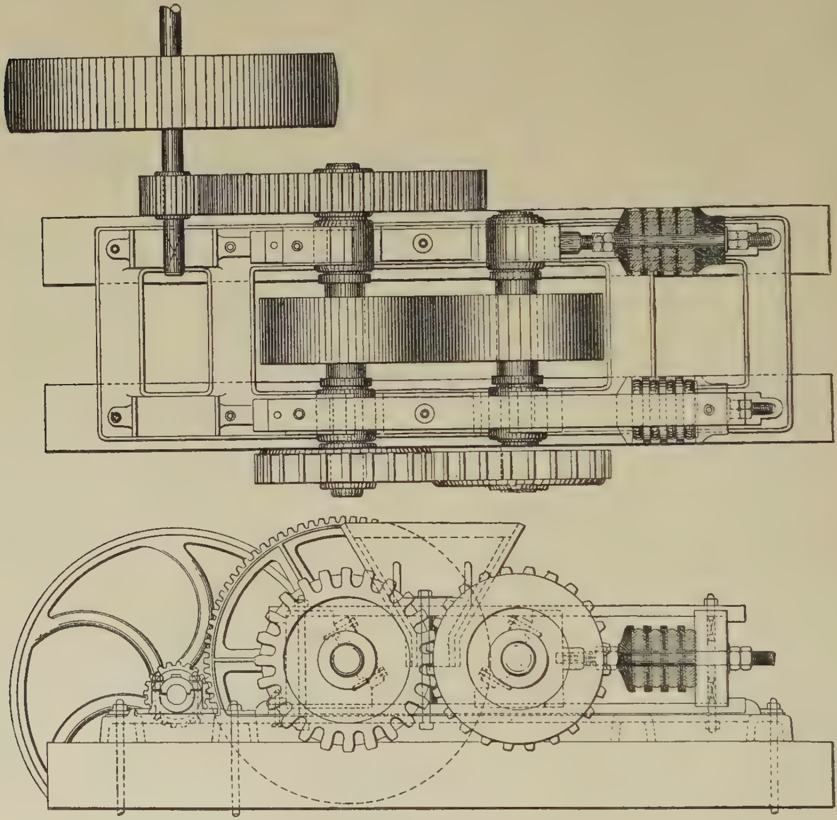


Fig. 30.

Geared Rolls.—Movable roll backed by rubber buffers under compression against standards rising from the bed-plate. The precise compression of these buffers is adjustable by nuts on a through-bolt whose backward or forward position is determined by the same nuts, with the result that the movable roll can be kept up to its work with the same force through all conditions of roller wear. It will be noticed that the movable block has a socket at its back into which the forward end of the through-bolt enters. By a cotter through this socket and through the bolt, the block is held, and the rollers are kept apart when no ore is passing. Distance-pieces to keep the blocks apart might then appear unnecessary, but such pieces are indicated in the illustration, their purpose being to eliminate all looseness and rattling. There is but a single driving pulley. The standards which take the crushing stresses are held together at the top by longitudinal tie-rods. A centrally disposed feed hopper is shown (pp. 57, 65).

simple and, on account of its slow speed, reliable ; moreover, the above objection loses force when it is considered that during crushing the movable roller is driven by friction from the other, the gearing only coming into action—to keep the movable roller up to speed—when little or no ore is passing.

Belted or Standard Rolls (Figs. 31, 37, 38).—The rollers of this type have much the same dimensions as geared rolls, though exceptionally the diameter may be as much as 60 in. or even 72 in. Both rollers being belt-driven, a normal and proper speed is 40—60 r.p.m. equivalent to 500—600 feet. At this higher speed a satisfactory capacity is obtained even when crushing finer than usual with geared rolls; by reason of their somewhat greater mechanical efficiency they effect this finer reduction at no greater power consumption.

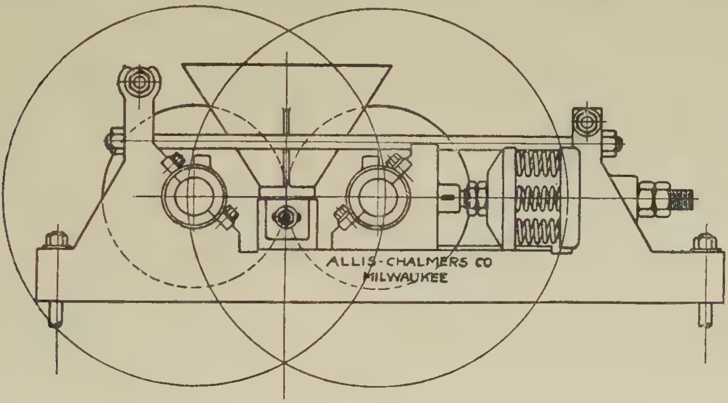


Fig. 31.

Belted Rolls.—Movable roller kept up to its work by helical springs taking their purchase against standards solid with the bed-plate. These springs are held between plates to make a nest. Through this nest, through plates and standard, a threaded bolt passes, its precise position being determined by forward nuts bearing against the spring plate, and by backward nuts bearing against the standard. By adjustment of these nuts the movable block can be given equal support wherever, throughout the limits determined by roller wear, it may be. No distance-pieces are shown in this illustration, the rollers apparently being in contact. Of the two equal-sized driving pulleys shown, that driving the fixed roller will be run with the tighter belt. In addition to longitudinal tie-rods there are also cross tie-rods (p. 59).

The capacity of these rolls is generally 10—16 tons per hour when crushing coarse material, or about 1000 lb. per h.p. hour; doing finer work the capacity would be proportionately lower.

Though, as mentioned above, both rollers are driven, the intention in driving the movable roller is only to keep it up to speed when little or no ore is passing. Actually, during crushing it is driven by friction from the fixed roller, any differential movement between the two being thereby reduced to a minimum.

The maintenance of the set distance between these rollers is obtained by the means indicated under geared rolls, these means being described in greater detail later (p. 63).

High-speed Rolls (Fig. 32).—Crushing finer still, the area of discharge being reduced owing to the narrow set, capacity can only be made good by increasing the speed, and high-speed rolls are employed. These have much the same diameter as before, but the speed is now about 100 r.p.m., equivalent generally to more than 1000 ft. at the periphery; exceptionally, speeds of 150 r.p.m. or even 200 r.p.m. are recorded.

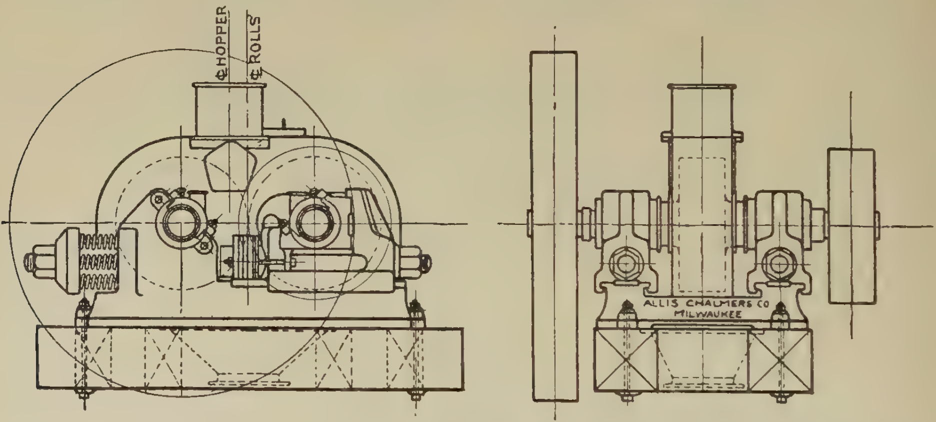


FIG. 32.

High-speed Rolls.—Movable rollers kept up to their work by tension rods which, while transmitting spring pressure, brace the machine, no additional tie-rods being necessary; this design is usual with high-speed rolls. Two driving pulleys of unequal size are shown, the larger one driving the fixed roller. The feed hopper, though not centrally disposed over the rollers, delivers a central stream. Around the rollers is a housing which terminates below in a discharge hopper. The great strength of the roller-shaft and the ample dimensions of its bearings are well indicated (p. 60).

Doing fine work, grooving as well as lack of parallelism between the faces would be fatal to efficiency, much of the material would pass uncrushed. In consequence, and a narrow width being much more easy to keep true, the faces of high-speed rolls are only 6—8 in. in width.

Respecting capacity, when crushing $\frac{1}{4}$ in. material to one millimetre, an ordinary high-speed roll consuming 20 to 30 h.p. will reduce 6—10 tons per hour, equivalent to about 600 lb. per h.p. hour. From this high capacity the drop is rapid as the faces become worn.

In these rolls, again, one roller is fixed and the other movable, this latter being kept up to its work by springs.

Rigid Rolls (Fig. 33).—In rolls of this type, apart from the means of adjustment, both rollers are fixed, that is to say, they are without springs and cannot give way. Such rigid rolls are only employed to reduce still finer material which is already fine, and in which accordingly there is little chance for unbreakable foreign material to be found; even so, a cheap breaking piece is inserted to protect the main organs. With both rollers immovable a more regular product results. Suitably to the fine crushing which is their province, these rolls have a narrow face and are run at a high speed. Being unable to force the rollers apart, the stream of ore passing between them is very thin, and the capacity of these rigid rolls in consequence is small. They are little used.

Other Roll Types.—Rolls for breaking weak material such as anthracite are generally toothed or corrugated; in addition, their width of face is greater in respect to diameter than ordinarily. It is quite exceptional to find corrugated rolls employed in metalliferous mining. Special corrugated rolls were designed by Edison for work at the Dunderland iron mine, Norway. In that design the rollers were several feet in diameter and not less in width of face, whereon, in addition to corrugations, were projections which acted as tremendous hammers, breaking the large lumps previously to passage.

DETAILS OF ROLL DESIGN

Rollers.—Rollers are invariably made of an inside core of iron and an outside shell or tire of suitable steel; usually the interior and slightly conical surface of the latter is drawn tight upon the similarly coned surface of the former, till no revolution of tire upon core is possible.

These two parts may be either single-coned or double-coned. Where they are single-coned, the core, a single and often hollow disc, is keyed to the shaft (Fig. 34). Upon it the shell, similarly coned, is held by longitudinal bolts seated in ways cut on the core surface, a lug at one end of the bolt engaging the shell while a nut at the other end engages the core.

Where the core is double-coned it is made in two separate halves, each a complete disc (Fig. 34). One of these two halves being shrunk or forced upon the shaft, over it the double-coned shell is brought, and then into its proper place the second half, the two smaller diameters facing one another with but little clearance between. Bolts through the two halves then bring them home, the result being a tightly-held but easily-released shell.

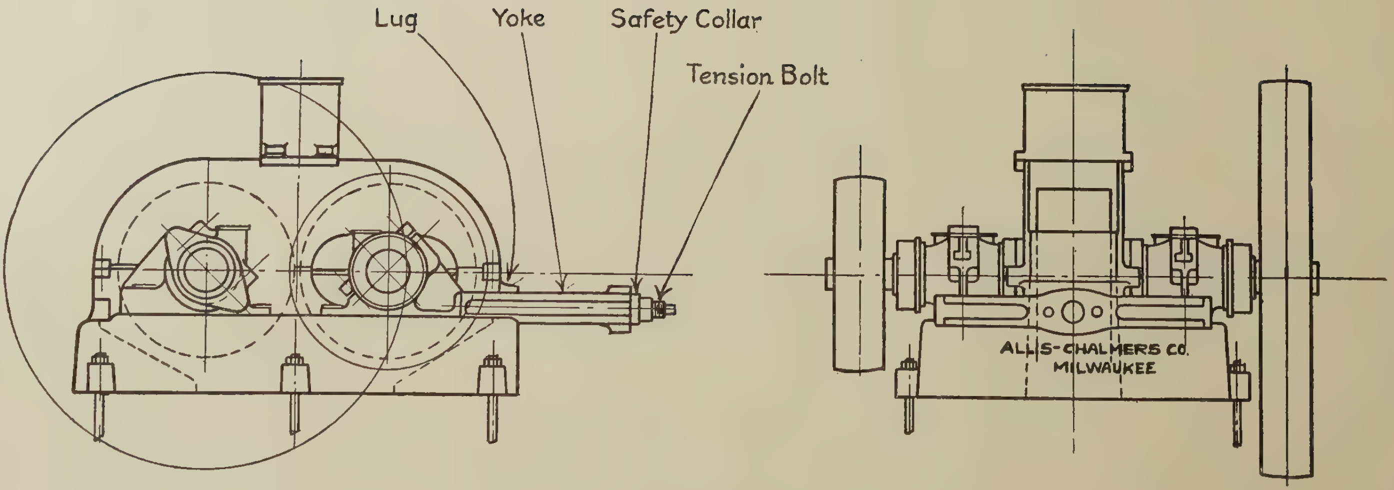


Fig. 33.

Rigid Rolls.—Adjustable roller maintained in position by the two arms of an embracing yoke, the position of this yoke being determined by a medial tension bolt. This tension bolt, while held in the yoke by a collar, screws into a lug on the main frame, the yoke being thereby advanced. The collar on the bolt serves as a cheap breaking-piece. Again, there are two driving pulleys of unequal size. The feed hopper is central (pp. 61, 67).

Sometimes the fixed half of a double-coned core has a central extension, upon which the second half sits, instead of sitting directly upon the shaft; this second half is then in the nature of a heavy split-ring, a design which possibly permits it to be drawn more snugly into the cone (Fig. 38).

In early types the core was often polygonal in shape, and the interior of the shell likewise; the polygon sides were then seats for wedges, wooden when crushing wet, steel when crushing dry.

Latterly some heavy shells, parallel and not coned, have been shrunk on the core after being heated; such shells must remain till they have worn sufficiently thin to be broken and detached by a blow. When the width of face is great, such a method of fixing the shell is facilitated by dividing it into two rings each of half the width.

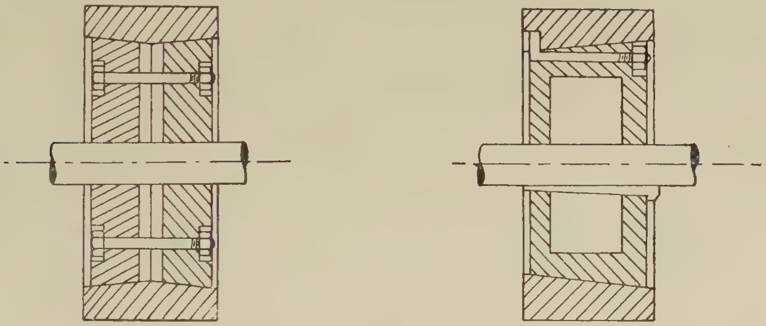


FIG. 34.

Details of Roller.—Core and Shell. Double-coned core with ordinary bolts; single-coned shell with special lug-bolts (p. 61).

With respect to shell dimensions, since the diameter largely determines the permissible size of feed, it should, at its maximum, be as large as possible. Diameter is, however, limited by the difficulty of uniformly forging shells of great size. In practice 54 in. is exceptional, 42 in. is uncommon, while 27 in. will be a fair average.

Concerning width, the stresses upon roller shafts and bearings increase so rapidly with increasing width that even when a true surface is not so essential, the width is almost always less than half the diameter, 24 in. being extreme and 15 in. an average width; with fine ore, for reasons already given, 6—8 in. is general.

Support of the Movable Roller and Adjustment of Set.—The movable roller may through its pedestals be forced up to its work by powerful springs behind it, or be drawn up by similar springs acting through tension rods. In either design it is necessary to provide: that the spring pressure shall not be permanently disturbed when the movable roller is moved forward as the roller faces wear; that when this adjustment of set is being made the

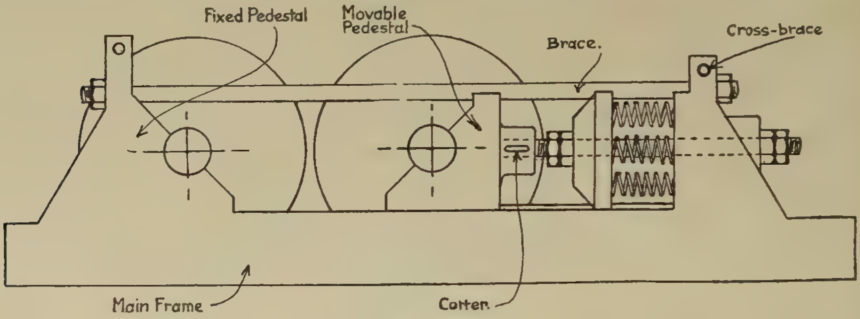


FIG. 35.

Simple Spring-rolls.—Adjustment of set as wear proceeds is made by first slacking away the two outside nuts, this allowing the springs to open and press the pedestal forward; then tightening the two inside nuts to bring the springs back to the same condition of compression. If it be desired to widen the set the nuts would be operated in a reverse sense, when the pedestal would be pulled back by the cotter which attaches it to the through-bolt. By slipping this cotter the pedestal goes free. When no ore is passing no spring pressure comes on the roller bearings, the whole of it being conveyed by the outside nuts to the frame. When ore is passing the pressure on the bearings is only that due to the resistance of the ore to crushing (p. 65).

spring pressure may be taken wholly off the pedestals for convenience in their manipulation; and that the full spring pressure shall only come upon

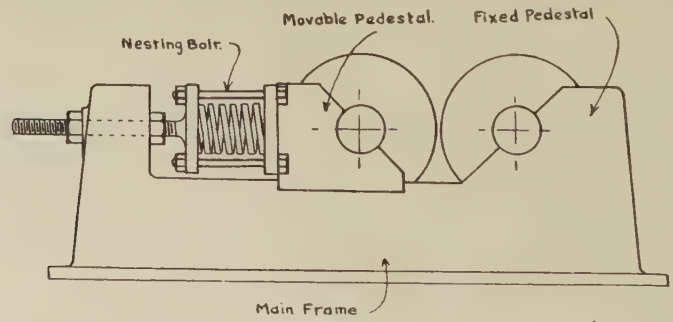


FIG. 36.

Simple Rolls—with Nesting Bolts.—Forward adjustment of the set is made by slacking the back nut and tightening the front nut, the nesting bolts preserving the spring pressure undisturbed. The forward nesting-plate is apparently directly connected to the movable pedestal (p. 65).

the pedestals—and consequently upon the bearings—under exceptional circumstances, as when the rollers are forced beyond their set to pass an unbreakable foreign body or a too abundant feed. The pressure ordinarily upon the bearing should be only that due to the crushing strength of normal material (Figs. 35, 37).

In the early designs and in the more simple of present designs the movable roller is forced up to its work, the fixed pedestal being either solid with the main frame or rigidly held in that frame (Fig. 35). To take up the outward thrust resulting from this design special braces are usually provided, though sometimes the frame is made substantial enough to do without (Fig. 36). In some rolls, and particularly with geared rolls, rubber springs take the place of the more usual steel springs (Fig. 30).

In the standard present design, however, the springs draw the movable roller up to its work by means of tension rods, one on either side, the machine not then needing any special braces. Each tension rod then passes successively through both pedestals on that side, the springs being either outside the fixed pedestal, which is more usual (Figs. 37, 38), or outside the movable pedestal.

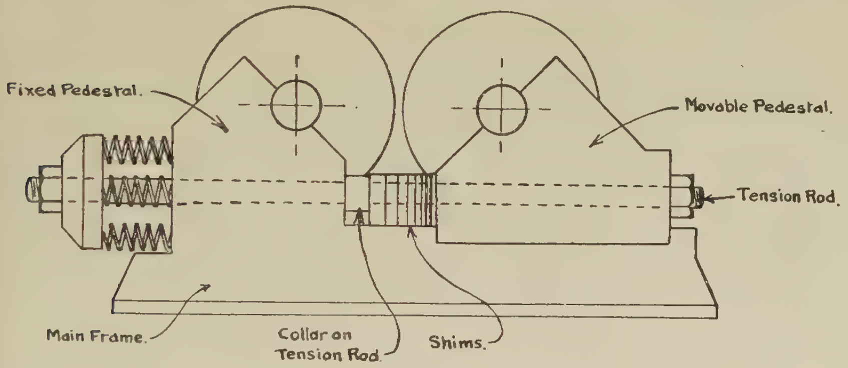


FIG. 37.

Standard Spring-rolls.—Diagram. The set is maintained as wear proceeds by slacking off the nut outside the movable pedestal, taking out a suitable shim, then tightening again. Throughout these movements the undisturbed condition of the springs is guaranteed by the undisturbed relative positions of springs and tension-rod collar. It is obvious also that the full spring pressure is transferred by that collar directly to the frame, the only pressure coming on the pedestal bearings being that due to crushing (pp. 59, 65).

In most machines the movable pedestal slides on the frame, its movement being guided by its long base. Exceptionally, however, it swings into position upon a heavy arm pivoted below (Fig. 39). In all designs embodying the tension rod the stationary pedestal is solid with the main frame.

Generally also the two movable pedestals, one on either side of the

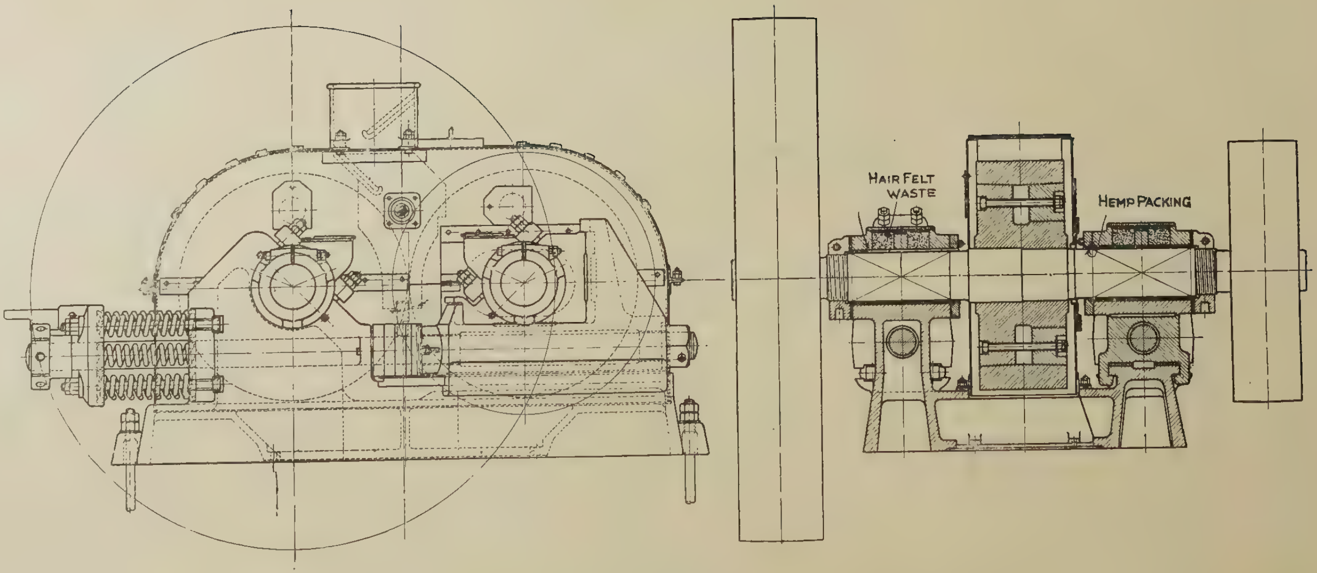


FIG. 38.

Modern Spring-rolls.—Side Elevation and Cross Section. The section is half through fixed-roller shaft and half through movable-roller shaft; the core is double-coned, the fixed half-core having an extension on to which the loose half is drawn. Feed-hopper and baffles shown in position upon the housing.

The set is closed to compensate for wear by slacking the latched nut so far that the required shim can be taken out, after which the movable roller is drawn up tight again. There is no collar upon the tension rod, but the springs are held undisturbed between plates by special nesting-bolts so arranged as to permit further compression of the springs when an unbreakable piece passes. This use of nested springs has the advantage that, when the machine is dismantled for attention or repair, the springs need not be released and the subsequent assembly of the machine is facilitated. Without collar, the tension rod can move freely either way. Springs and adjusting nuts are all at the fixed end (pp. 59, 63, 65, 69).

machine, are independent, that is to say, though there may be some means of moving them together, they are not rigidly connected. In some designs, however, and particularly for very fine crushing, these two pedestals, well guided at their base, form part with a heavy yoke which extends round the back of the movable roller, the parallel advance of the movable roller being thereby perhaps better assured. Such a design is chiefly employed with rigid rolls (Fig. 33).

This parallelism of the two rollers is of prime importance not only to the smooth running of the machine, but also to the regular quality and amount of the product. When distance pieces are used, it is obtained by drawing the movable pedestal tightly against the same number of such pieces on either side of the machine. Without distance pieces, the two adjusting nuts, one on either side, must be advanced or retired equally. These nuts are usually moved independently, but, as great care is then demanded, in some "synchronized" rolls they are moved in unison, by making each nut a small worm-wheel to be engaged by one of two worms upon a single spindle (Figs. 40, 41).

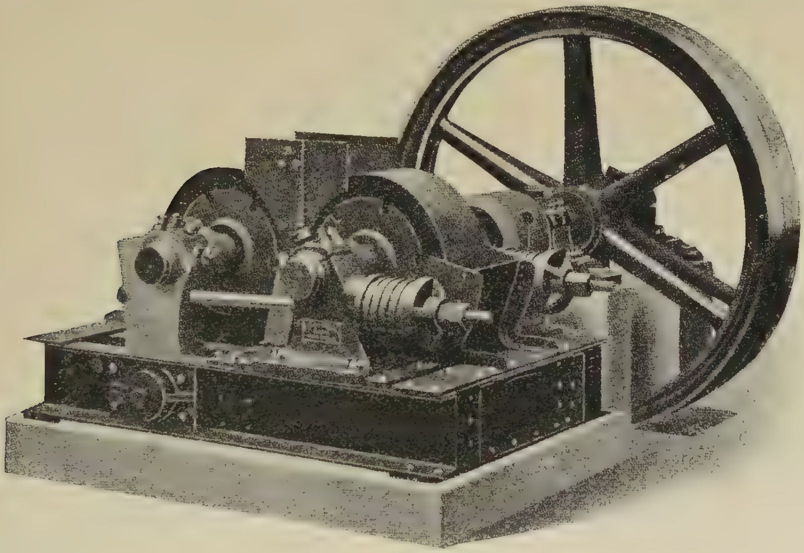


Fig 39.

Swinging-arm Rolls.—The illustration clearly indicates how the movable roller is brought and held in position. It further shows the lugged ends of the bolts securing the shell to the roller core ; and illustrates a girder bed-plate (p. 65).

Much obviously depends upon the nuts. When these are worm-wheels engaging worms they cannot slacken. Otherwise, and more

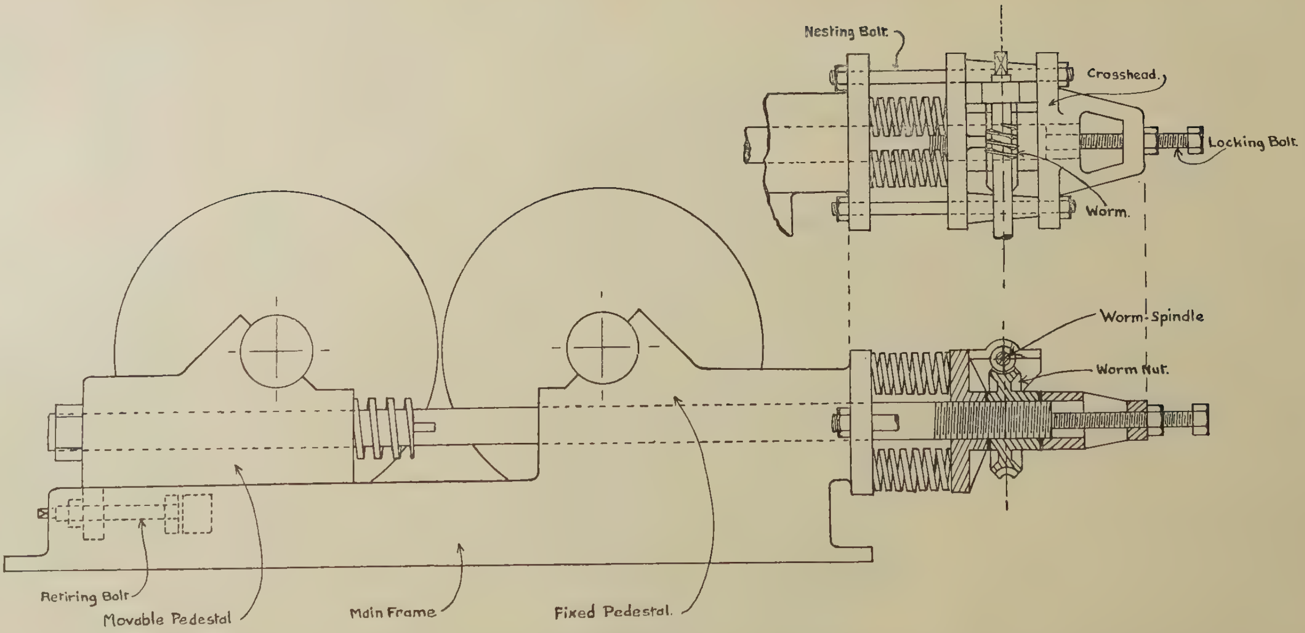


Fig. 40.

Synchronized Rolls (Wilfley).—Outline Drawings. The nuts on the tension rods being worm-wheels are moved in unison by two worms on the same spindle. To draw the rolls nearer together the locking bolt at one end and the retiring bolt at the other end are slackened to the desired extent. That being done, the movable roller is brought forward by turning the worm-nuts, after which the tension rod is again secured by the locking bolt. To open out the rolls, the worm-nuts are turned in the reverse direction, whereby, and because these nuts are held against an outboard cross-head, the tension rod is retired, but not necessarily the movable pedestal. An interior spring would tend to move that pedestal, but, unless the rolls were running and consequently alive, it would be unable to do so. To bring the movable pedestal back there is a retiring bolt at the back end of the machine, this bolt passing through a threaded lug depending from the movable pedestal. The interior spring mentioned above is to prevent the rolls hurtfully striking together after having been forced apart at the passage of an unbreakable piece (pp. 67, 69).

commonly, additional or special lock-nuts are used, or the nut is held by a latch (Fig. 38). Equally, the tension rod must not be free to rotate, this being attained either by giving it an angular section at some point along its length, or by holding the bolt head in a suitable seat.

The amount of movement of which the movable pedestal shall be capable depends largely upon the permissible wear of the roller shells. With tires 5 in. thick, a wear of 4 in. on either roller must be accommodated, or 8 in. in all, to which figure the desired range of set may add another inch. Generally, however, thinner shells than these are used, and less movement suffices.

Where distance pieces are employed adjustment of the set requires the machine to be stopped. That also is the general practice. Where, however, worm nuts are used it is generally possible to adjust while running, whether it be desired to close or to open-out the rolls. The latter movement is generally left for the ore to perform, though sometimes the assistance of auxiliary springs is invoked (Fig. 40).

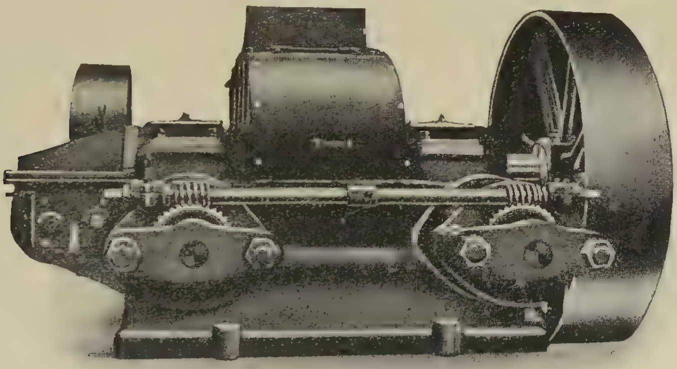


FIG. 41.

Synchronized Rolls (Traylor).—General View. This illustration presents the synchronizing gear very clearly. It shows the worm-wheel nut held by a cross-head, outboard; it illustrates also the generous length of the roll-shaft bearings (p. 67).

Spring Pressure.—Ordinarily the rock submitted to roll crushing, shattered by previous blasting and breaking, will have a maximum strength of about 6000 lb. per sq. in., when, for single pieces as large as 2 in., a crushing force of 20,000 lb. would be necessary. The total spring pressure provided is generally from 20,000 lb. in small machines to 150,000 lb. in large machines. *Other Details of Design.*—The bearings must be wide and dust-proof; the movable bearing is sometimes provided with a swivel seat, to prevent undue stress at passage to one side of unbreakable material. To confine small pieces and dust, housing is necessary, particularly with the finer rolls. Cheek plates are required to keep the material from escape at the sides of the crushing throat. Roller shafting must be amply large and strong to be perfectly stiff and rigid.

Wear of Roll Shells (Fig. 42).—Owing to the abrasive nature of ore, wear of the roller faces is unavoidable. Moreover, with the greatest care such wear is rarely even, grooves gradually develop at the centre and flanges at the side.

Grooving results from the tendency of an ore stream to flow fastest and deepest down the centre, as well as from slip and from differential running between the two rollers; it is destructive of good work, particularly in fine crushing, since with grooving developed much of the material passes uncrushed. Flanging results from bad alignment, the two rollers not running accurately on one another but leaving untrodden the end of one roller and the opposite end of the other. Flanging causes end thrust.

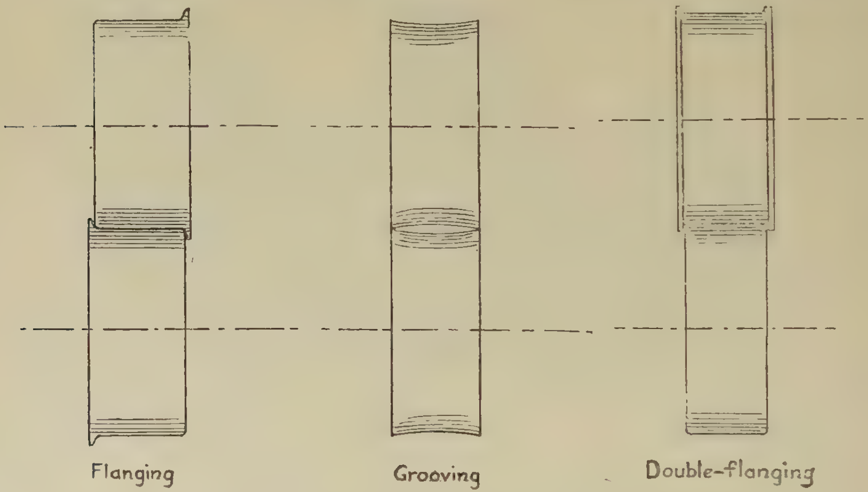


FIG. 42.

Wear of Roll Shells.—Flanging; grooving; double-flanging (p. 70).

It was usual, formerly, to take out the rollers when grooves or flanges had seriously developed, and to turn them true in the workshop, but, sometimes now, the rollers are ground in place and while the machine is running, by means of an emery wheel fixed on the frame. Or the fixed roll, then called the fleeting roll, is given an imperceptibly-slow lateral movement of about half an inch from side to side, this being sufficient to wear down flanges and to moderate grooving.

Exceptionally, the contrary idea is followed and flanging is encouraged. Then, one roller is made a little wider than the other; sometimes indeed it is provided with rudimentary flanges to lead and confine the narrow roller. This practice is seen at Broken Hill, New South Wales, and at Joplin, Missouri. By it end thrust is diminished and a less massive frame suffices. It has, however, not been adopted for heavy work.

In amount, wear depends upon the relative hardnesses of ore and shell. Shells are 2—5 in. thick when new, and $\frac{1}{2}$—$\frac{3}{4}$ in. at discard, the gross amount of wear varying from 0·01 to 0·1 lb. per ton crushed. Manganese steel, chrome steel, forged steel, and cast-iron are all used, and a heavy shell of suitable material might in its life take part in the coarse crushing of one hundred thousand tons.

Practice in Roll Crushing

Ratio of Size-reduction.—With breakers, it was seen, a size-reduction of 5 was common, though 4 might be better. With rolls 4 is common, though 3 would be better, these ratios being those obtaining between the maximum size of feed and the set of the machine. The finished product of a roll is, however, generally determined not by the set but by the aperture of a sizing appliance in " closed " circuit with the roll; that aperture, it is true, is usually the same as the set, but even so a good deal of oversize results from the freedom with which flat pieces pass through rolls. Exceptionally, the screen aperture is much smaller than the set distance between the rollers. At Broken Hill, for instance, the rolls are set about $\frac{1}{2}$ inch apart while the screen working with them has a $\frac{1}{8}$ in. aperture, the material above that size being returned to the rolls for further crushing. In that way a large ratio of size-reduction, namely, about 12 : 1, is obtained by one machine, though at the cost of handling much oversize (Fig. 48).

Large-diameter rolls give greater possibilities of reduction than those of smaller diameter. Large rolls are also necessary with large material in order that the wedge angle may remain within the angle of nip. With great diameter, however, great stresses arise, and uniform shells are more difficult to forge, so that it often becomes preferable to do in series and by smaller rolls what otherwise might be done by one pair of larger rolls. Giving it a wider set, the first member of that series, in spite of its smaller diameter, would still be able to nip large material.

Stage Crushing.—To crush in a series of operations, removing the finished material resulting from each operation, constitutes stage crushing.

Rolls are often placed to take the product from breakers, that is material of $1\frac{1}{2}$—$2\frac{1}{2}$ in. ring. Such rolls, set to about $\frac{1}{2}$ in. aperture, are spoken of as Coarse or Roughing Rolls; they would be of relatively large diameter and slow speed. Following these would come the Fine or Finishing Rolls which would have a somewhat smaller diameter but, making a greater number of revolutions per minute, would have a higher peripheral speed. If the finishing work itself were done in two stages, the two sets of rolls necessary would generally have the same diameter, though the final stage would be run at a higher speed. The advantage of such an arrangement is that the shells on the final rolls, when so grooved as no longer to be efficient, could still be used on the

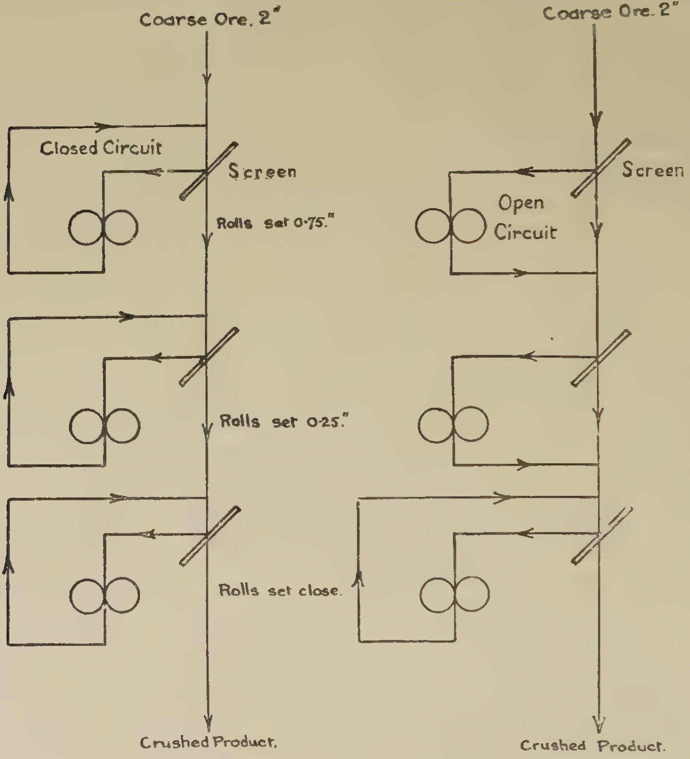


FIG. 43.

Stage-crushing, employing Rolls.—Diagrammatic Representation. In the left-hand diagram the oversize is returned at each stage, this necessitating considerable intermediate elevation of oversize. In the right-hand diagram only the finishing rolls are worked in such a closed circuit, this being simpler and requiring less elevation. The screen working in circuit with a set of rolls generally has an aperture equal to the set distance between the two rollers (pp. 57, 71).

Medium rolls. A typically complete sequence might be represented by the following:

	Feed.	Set.	Diam.	r.p.m.
	In.	In.	In.	
Coarse or Roughing Rolls .	2	0·75	36	40
Medium Rolls	0·75	0·25	27	70
Fine or Finishing Rolls . .	0·25	0·05	27	100

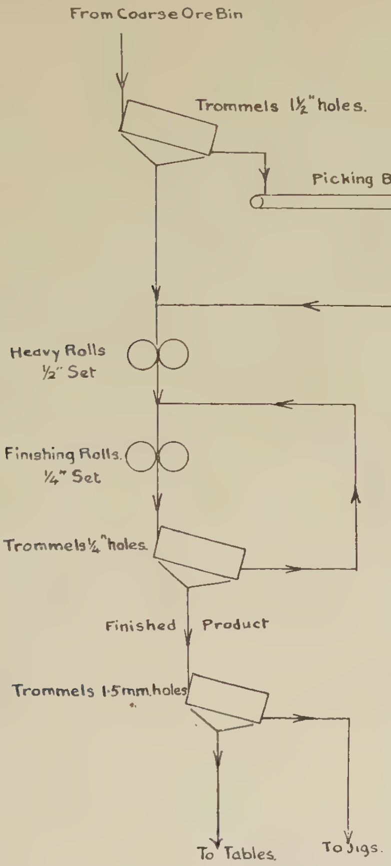


FIG. 44.

Breaking and Roll-crushing at the Britannia, British Columbia (1916).—The ore consists of chalcopyrite disseminated in large lenticular ore-bodies in a big mineralized zone. It is low grade, and concentration is complicated by the presence of sphalerite and pyrite. The tailing from the jigs is further crushed by fine rolls set to 2 mm., then concentrated by flotation. It will be noted that only the finishing rolls are worked in closed circuit (p. 79).

With stage crushing each succeeding machine has less to crush, while that portion of the crude ore which as delivered from the mine was already fine enough, is kept out of the crushing machines altogether

(Fig. 43). Where, therefore, the scale of operations is sufficiently large that the decreasing amounts can be accommodated by employing a smaller number of standard-size machines in each succeeding stage, stage crushing is economical in power consumption (Figs. 46, 47, 49). In other circumstances and to avoid complication, single-stage crushing is preferred, though at the risk of a circuit congested with oversize.

Arrested and Choke Crushing.—As will be seen later, stage crushing may be practised with all sorts of crushing appliances but with none so clearly and definitely as with rolls, because with them crushing is arrested and complete pulverization does not take place, the roll normally having no power to crush smaller than its set. Such crushing is spoken of as "Arrested Crushing" in contradistinction to that more complete disintegration which takes place when, as with stamps for instance, no set is interposed. Arrested crushing permits crushing to be more closely associated with screening, and stage crushing to be more definite in consequence.

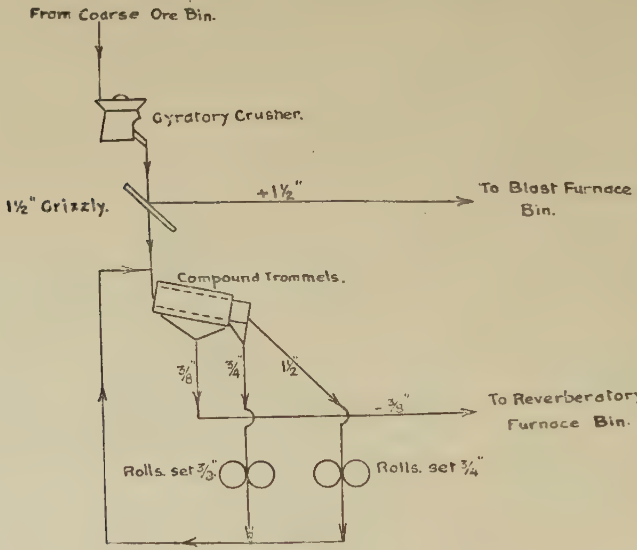


Fig. 45.

Breaking and Roll-crushing at the United Verde, Arizona (1917).—In preparation for smelting. The ore is predominantly chalcopyrite with other subordinate sulphides; it carries excess of iron and sulphur (p. 79).

This condition of arrested crushing is, however, only completely fulfilled when the feed is at such a rate that in the crushing throat there is little interference between individual pieces, and the discharge is free. The amount of material then passing through the machine is much less than that which would constitute the hypothetical ribbon outlined by the dis-

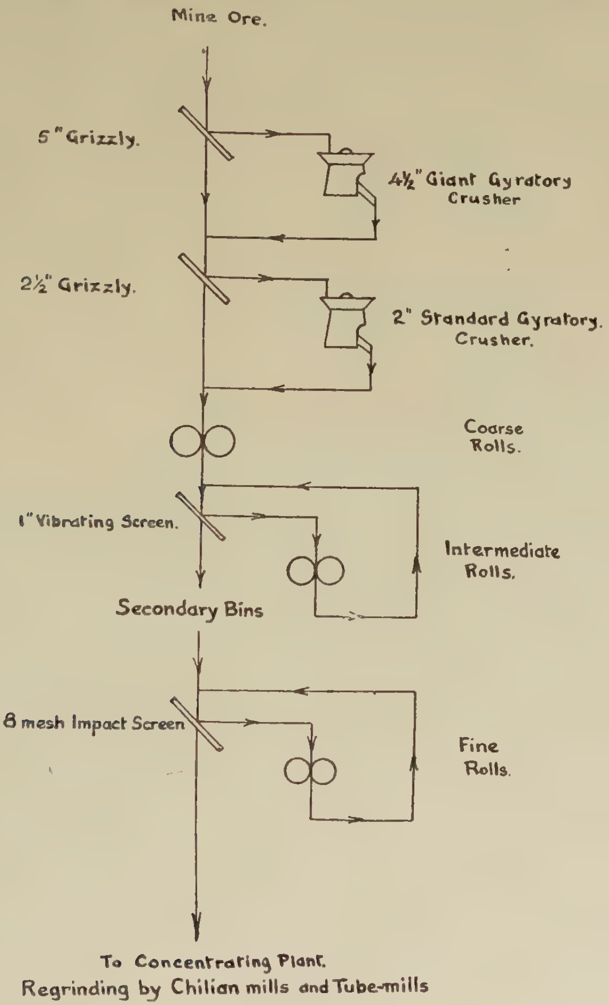


FIG. 46.

Breaking and Roll-crushing at the Utah Copper, Arizona (1919).—Low-grade, disseminated copper ore; chalcocite, covellite, and chalcopyrite disseminated with pyrite in monzonite and in quartzite. Gigantic installation. The intermediate and the fine rolls are worked in closed circuit with screens. Apart from regrinding, five stages in comminution are illustrated (pp. 74, 79, 656).

charge area and the peripheral speed. Broken ore contains voids and on that account not more than 60 per cent of the hypothetical ribbon could possibly be passed through a given and unaltered set. The limitations of feeding devices do not permit even that figure to be reached without occasional overfeeding, and 30 per cent is as high as can be maintained in practice, 20 per cent being more common. That is to say, feeding with every desire to feed fully but not to choke, the weight of ore crushed is generally about 20 per cent of that of the volume represented by the

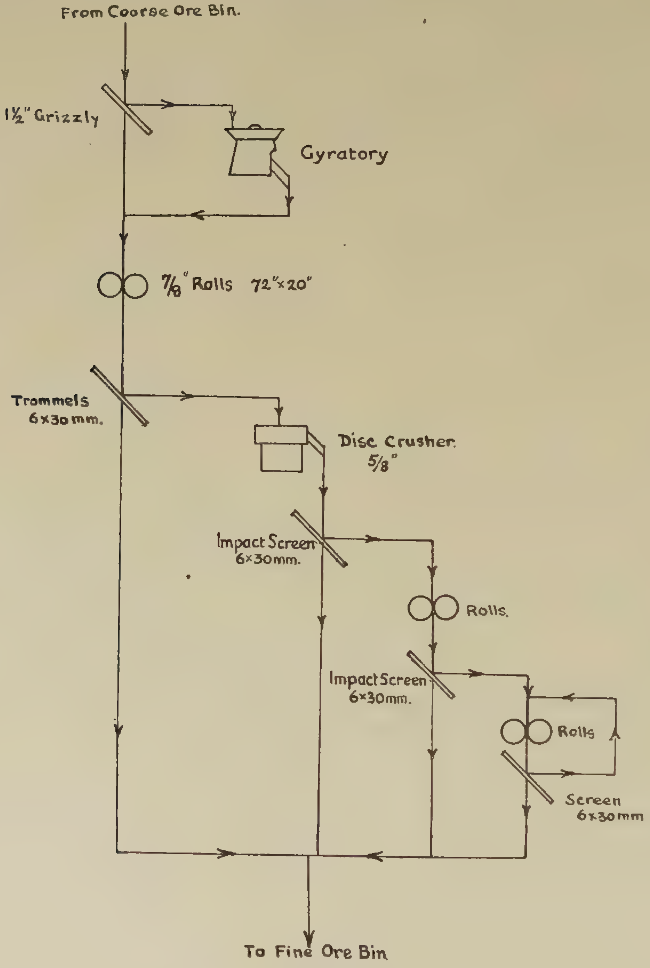


FIG. 47.

Breaking and Roll-crushing at the Braden, Chili (1915).—Low-grade, disseminated copper ore, a shattered andesite carrying chalcopyrite, bornite, and chalcocite. In this flow-sheet five stages in crushing are illustrated, of which only the last works in closed circuit. After the fine ore-bin comes regrinding (pp. 53, 74, 79).

product of speed and discharge area. Calculations based upon that percentage give reliable figures for the capacities of rolls generally.

When, however, the ore is fed at such a rate that the set of the machine is insufficient to pass the amount seized, the rollers are forced apart and the material moves forward, not freely, but under great pressure and in a state of congestion or choke, crushing being largely accomplished by the particles among themselves. Such conditions constitute "Choke Crushing" or "Choke Feeding." Under them the product bears a less definite ratio to the set of the machine than does the product from free crushing, and is less granular. Choke feeding is accordingly not rational in stage crushing. It is only used designedly in fine crushing, and then because of the somewhat greater capacity it gives the machines. In choke feeding, though less wear comes upon the shells, undesirable pressure comes upon the bearings.

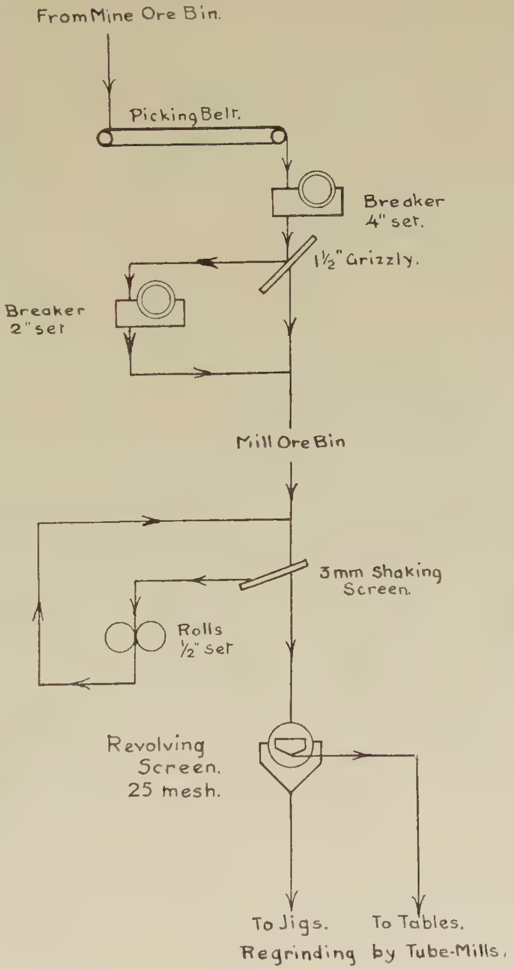


Fig. 48.

Breaking and Roll-crushing at Broken Hill, N.S.W. (1918).—Complex lead-zinc-silver ore. Stage-crushing by rolls was here given up; single-stage roll crushing now reduces the breaker product to pass a 3-mm. screen, not in one passage, but by continual return of the oversize (pp. 58, 71, 79, 656).

Roll Feeding.—Additional to the observations already made in respect to feeding, one or two other points deserve mention. The ore must be fed regularly or otherwise the power requirements will be irregular, and hurtful vibrations will ensue. Moreover, the arrival of much material at irregular intervals would engender grooving. In coarse crushing this would not be so great a point, and the convenience of taking the product direct from the breaker would probably outweigh the disadvantage of a somewhat irregular feed. But with fine rolls a regular and continuous feed is a necessity, and such rolls are usually provided with a special feeder; if then the feed arrives by a tray it may be better to have the bottom of that tray convex upwards, in order to keep the stream from concentrating at the middle.

Roll feed is almost invariably dry; wet roll-crushing is only practised when fine crushing is attempted, the reason being that it is easier to screen fine material when water-borne, than to screen it dry. A little water is, however, often directed on to the roll faces to keep them in good condition for seizure.

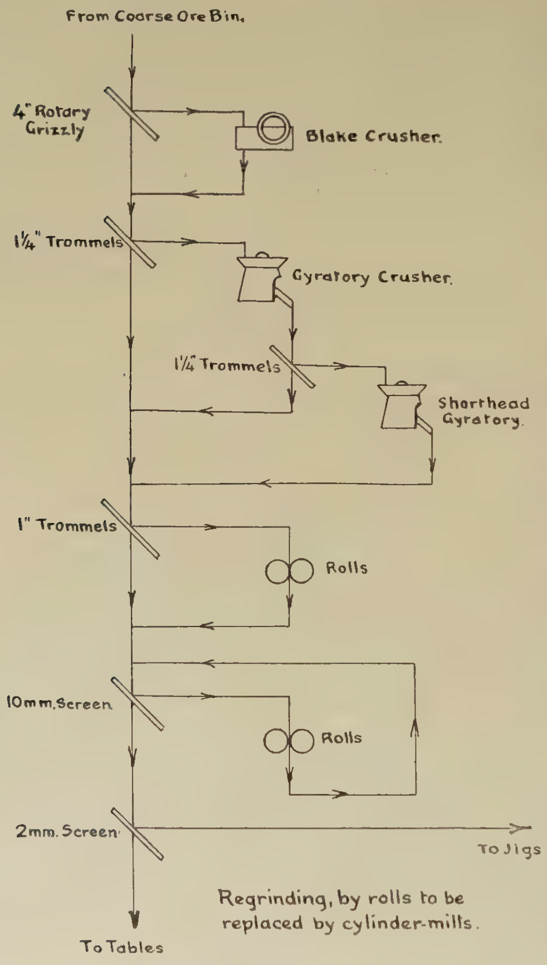


FIG. 49.

Breaking and Roll-crushing by the St. Louis Smelting and Refining Company, Missouri (1917).—Disseminated lead ore assaying 4 to 6 per cent of lead; ore-mineral, galena; gangue-mineral, dolomite. Large installation; breaking in three stages; roll-crushing—excluding rolls used for regrinding middling-products—in two stages; fine rolls working in closed circuit; all the other machines in open circuit (pp. 50, 74, 79).

Application. — Rolls are good intermediate crushers where the material is not too hard. They have large capacity, low operating-cost, and mechanical simplicity. Applying arrested crushing they are good granulators, and their use with jigs and tables in the water concentration of base-metal ores is wide (Figs. 44—50). For finer work their capacity falls off greatly, and their range of application may be said ordinarily to lie between 2 and 12 in. mesh. Beyond this, rolls to-day give place to more suitable machines, and particularly to a granulating type of the cylinder mill.

Rolls have been little used in the crushing of clean precious-metal ores for hydro-metallurgical treatment, these ores being largely quartzose and generally too hard.

The cost of a limited crushing by rolls is low and generally about 6d. per ton, but where complete comminution is accomplished that figure will require to be proportionally multiplied.

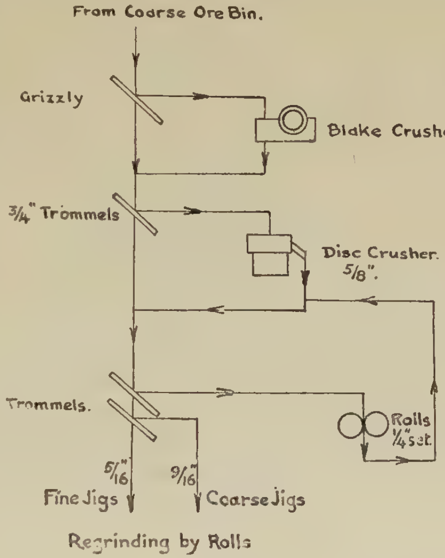


Fig. 50.

Breaking and Roll-crushing at the Thomson Zinc Mine, Wisconsin (1917).—Ore consists of blende mixed with galena, in limestone and shale (pp. 53, 79).

Pendulum Mills

In these mills, rollers swinging by their spindles from bearings above, run around the inner surface of a fixed ring-die, against which they are flung by the centrifugal force derived from their gyration around the central axis of the machine, while they themselves rotate around their more or less vertical spindles (Figs. 51, 52). As with rolls, with these machines also, there exist two circular surfaces between which crushing takes place, while a further point of similarity exists in that with each type of machine the movement of the roller is straight forward, and always in the same plane. With rolls, however, two parallel and equally-limited faces roll one on the other in the same vertical plane, each providing a never-ending path for the other, whereas with pendulum mills one circular surface is extensive and stationary while the other is limited and moving, the inside face of the former constituting the path for the latter, the plane of movement being horizontal. Moreover, in these machines gravity does not assist the entry of the ore into the crushing throat but tends contrariwise; the ore is thrown into the path of the oncoming roller by centrifugal force, gravity acting against an even distribution over the vertical crushing face. For this reason the face is narrower with this type of machine than with any other, being generally 6 in. Further, the die, curving inwards around the roller, makes with the surface of that roller a less open crushing-throat than characterizes the rolls. Similar considerations, however, apply in both cases; if the wedge angle between the two tangents drawn from the points of contact with the ring-die and roller surface, respectively, be less than the angle of nip, the roller will ride upon the material and exert due pressure, whereas if it be greater the roller will tend to force the material around the

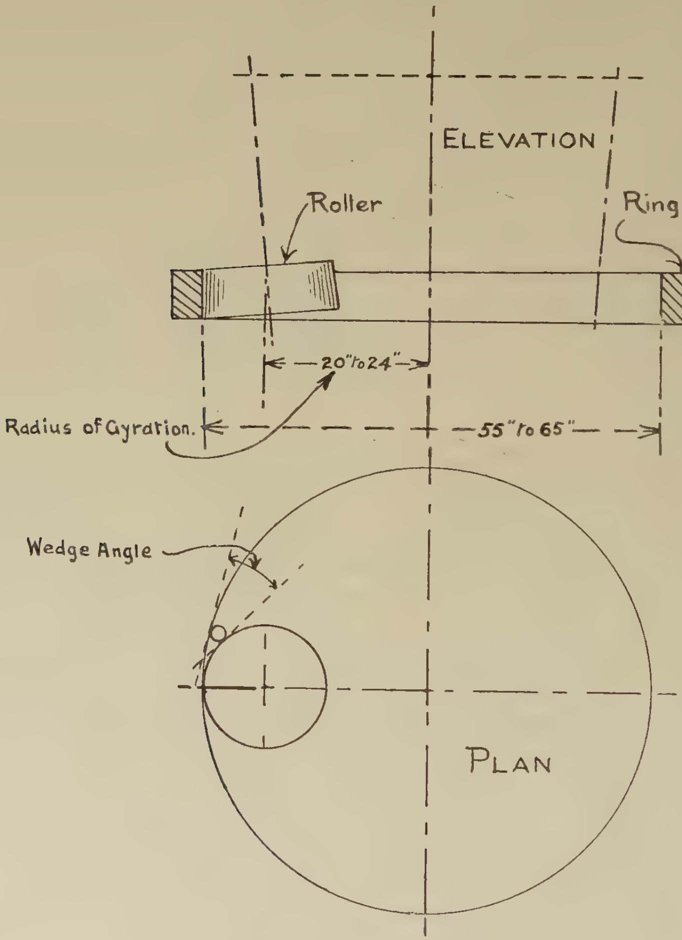


Fig. 51.

Huntington Mill.—Diagram of crushing surfaces (p. 79).

ring. This latter, of course, must not happen or the machine would soon be choked, and these mills must in consequence be fed with a suitably fine material. Moreover, with larger pieces upon which the rollers might ride and yet be ineffective, a hurtful vibration would develop.

There are two principal mills of this type, the Huntington mill and the Griffin mill.

Huntington Mill (Fig. 53).—In this mill the ring die, 2—3 in. thick, 6—8 in. deep, 50—60 in. internal diameter, is set in a pan-casting, wherein it is fixed by wedges regularly disposed around its circumference. Outside, on the bottom of this pan, is a heavy flange by which the machine is bolted to its foundations, while, circumferentially around, comes a gutter to collect and lead the crushed product to the discharge lip at the front.

Upon this pan is bolted the housing, in the lower portion of which —and thus not far above the ring die—is situated the screen discharge, which occupies most of the circumference for a depth of about 8 in. Above this screen the housing continues in order to contain the splash, the total height from base to upper rim being thus about 3 ft.

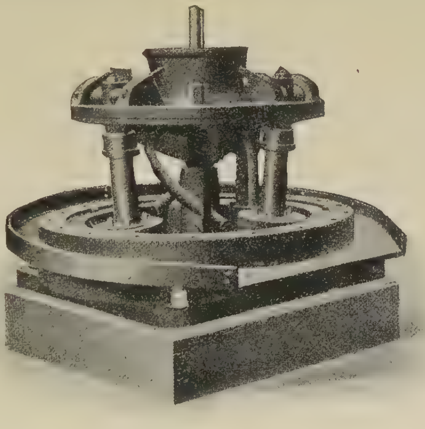


Fig. 52.

Pendulum Mill.—Modern Type. Three rollers; overhead horizontal driving pulley (not shown); central feed; housing removed to show rollers, ring die, feed pipes, etc. (p. 79).

At the centre the bottom of the pan rises steeply to form a passage for the driving shaft of the machine. On to this shaft is keyed the spider from which the rollers hang. To obtain the necessary capacity and at the same time a well-balanced machine, there are usually four of these rollers. Each roller consists of the roller proper, the spindle on which the roller turns, and the sleeve which holds the spindle (Fig. 54). At the top this sleeve spreads horizontally to form two trunnions, these latter sitting in bearings so disposed on the spider that the roller is free to swing radially in respect to the central axis. Even when at rest—the spindle being a little overhung—the roller hugs the die, a position which it holds with crushing force when working. In this overhung position the roller is a little canted, so that its clearance from the pan bottom, which at the die is only $\frac{1}{4}$ in., is increased to 1 in. inside. Covering the pan bottom is an annular liner.

The roller proper is 17—20 in. in diameter and, like the die, 6—8 in. deep. It consists of an outside tire generally of steel and a cast-iron core extended to embrace the steel spindle, the weight of the core and tire together being 450—600 lb.

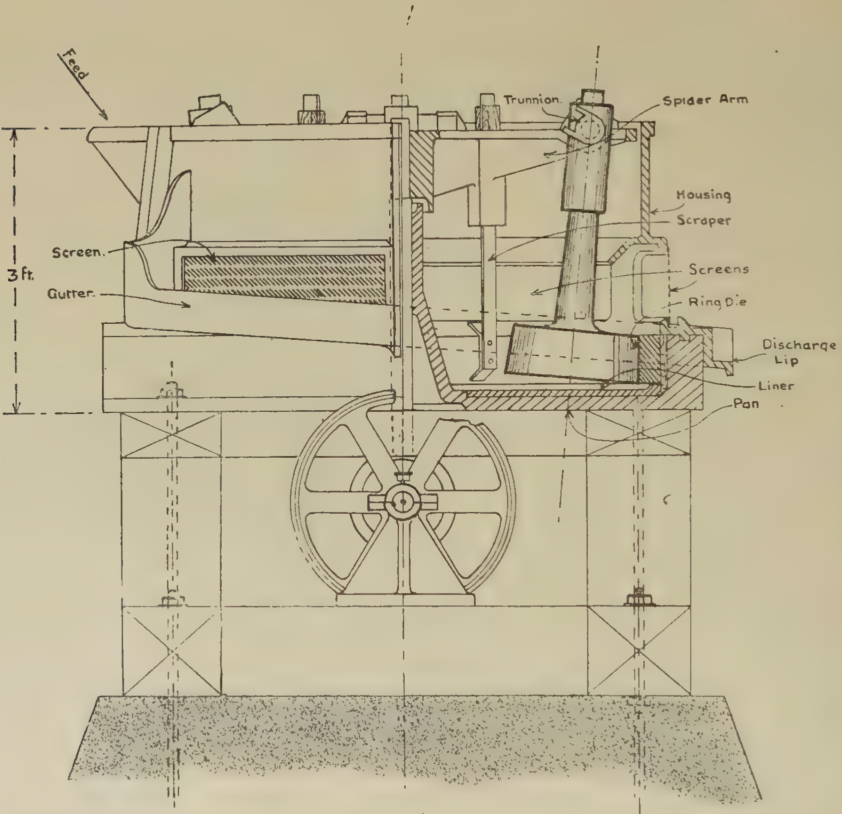


Fig. 53.

Huntington Mill.—General Drawing (p. 81).

So constituted, the machine is driven from the central shaft, which is rotated either by a horizontal pulley above, this being the most direct and efficient way, or by means of bevel wheel and pinion beneath, which is often the more convenient way. The speed is generally from 60 r.p.m. in the large size of mill and with relatively coarse ore, to 80 r.p.m. in the small size and with fine ore, the roller at the same time making about 270 r.p.m. on its spindle.

The resultant crushing pressure of this movement is that of the centrifugal force developed, obtainable from the formula:

$$F = \frac{Mv^2}{r} = \frac{Wv^2}{gr} = \frac{Wn^2\pi^2d^2}{gr},$$

where W = weight of roller, r = radius of gyration, and n = number of gyrations per second. Of these factors, weight and number have already been stated; it remains to give the radius of gyration. This radius is that of the circle described by the lower end of the roller spindle. It is ordinarily 20—24 in. From these figures it may be calculated that, apart from some impact developed, the crushing force is about 2000 lb., a figure indicating that this machine is not capable of crushing large material; in practice $\frac{3}{4}$ in. is about the maximum size.

With respect to the introduction of the feed, in the earlier types the ore was fed at the periphery down a chute forming part of the upper housing. In this manner, though the material arrived fairly into the crushing space, its distribution around the whole circle was not adequately secured, but depended upon the sweep of the rollers themselves and upon the action of scrapers set behind each of them. The central feed, now more usually seen, is in this respect much more satisfactory and the machine runs smoother; with it, the feed chute instead of being attached to the housing is an annular cup upon the spider frame, the material fed into the cup passing down into the pan by pipes. With these pendulum mills generally, a separate feeder independently driven is employed. If underfed they

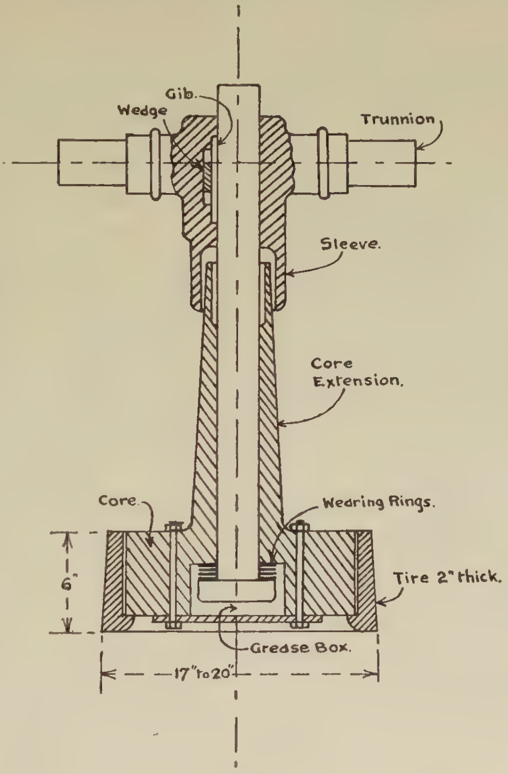


FIG. 54.

Huntington Mill.—Details of roller (p. 81).

are noisy and rattle, if overfed they slow down and throw off the belt.

Concerning discharge, it must be remembered that in one very important respect this machine differs from rolls, namely, that the crushing is not arrested but may proceed to complete pulverization. Such complete pulverization not being ordinarily required, screens must be provided whereby the material crushed fine enough may escape. Screens thus become an essential part of the machine.

These screens, as already stated, are set closely above the die, practically all around the circumference. To raise the fine material to this discharge, water at the rate of about 6 tons of water to 1 ton of ore is necessary. This water is entered with the feed, assisting in its distribution. Carried to the screens by the circumferential swirl, the crushed material finds exit with the water; but, the apertures being foreshortened to its passage, the particles have considerably less chance to get through than had they struck the screen normally. The screens of this machine, consequently, are liable to become blinded, though this is largely mitigated if slotted or rectangular apertures be used and these be placed with their long dimension in the direction of the swirl, at an angle of about 45° to the horizontal.

The wear of crushing comes on the ring die and on the roller tire, the former wearing from an initial thickness of 2 in. to a final $\frac{3}{4}$ in. in about seven weeks, when it is discarded, while the latter wears from an initial $1\frac{1}{2}$ in. to $\frac{1}{2}$ in. in four or five weeks. With uneven wear the capacity of the machine becomes reduced and vibration becomes intense. To return the tires to good condition they are taken out and machined, while to recondition the die a special shoe shaped to the curvature is introduced in the place of one of the rollers; being unable to turn, this shoe grinds the die face. Uneven wear is sometimes so pronounced that the tires discarded weigh 40 per cent of their original weight.

In addition to wear, a good deal of repair becomes necessary by reason of vibration, and the maintenance cost is high. There are, however, several good points about these machines. Their prime cost is relatively low, they require little height, are easily set up, while in relation to capacity their total weight is low and the cost for transport correspondingly small.

With respect to operating cost, apart from maintenance the items are not high, and the inclusive cost is usually less than one shilling per ton. Owing to the generous screening area, discharge is effected early and the capacity is high, while the product is granular and in good condition for mechanical concentration.

From the foregoing it will be realised that Huntington mills have but a limited range of application. They are satisfactory for softish material not greater than $\frac{3}{4}$ in., and particularly when water concentration follows, since their product is granular; but even for such work they are being replaced by cylinder mills of a type designed to give a granular product. These machines have also been used to crush soft gold ores after fine breaking, in such work proving themselves good amalgamators. They are usually made in two sizes, known respectively as the 5 ft. and the 6 ft. mills. With ore not too hard the former will crush 20 tons per day from $\frac{3}{4}$ in. to pass 16 mesh, that is, a reduction of 24 : 1, at a consumption of about 12 h.p., these figures being equivalent to about 166 lb. per h.p. hour. The capacity of a large machine crushing $\frac{1}{4}$ in. material might very well be 60 tons per day.

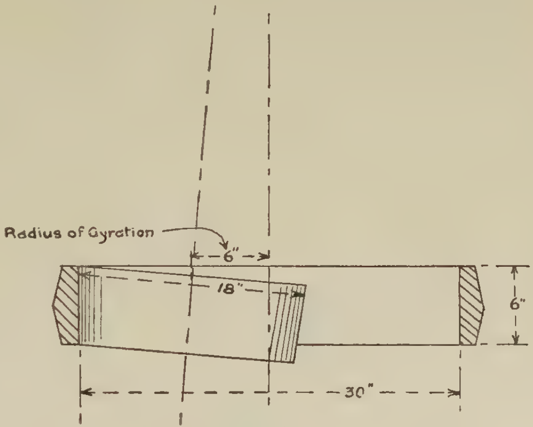


FIG. 55.

Griffin Mill.—Diagram of crushing surfaces (p. 85).

Griffin Mill (Figs. 55, 56).—This machine works similarly to the Huntington mill, though having but one roller it differs a little in design. A single roller hangs on a spindle which ends above in a universal joint within a driving pulley revolved horizontally. The tire of this roller is about 18 in. diameter, 6 in. deep, and weighs about 120 lb., while the total weight of the roller is about 600 lb. The fixed die is 6 in. deep, the same as the roller, 30 in. internal diameter, and weighs about 300 lb. When at rest, the roller spindle hangs in the vertical axis of the machine, and the roller concentrically within but not in contact with the die circle. At starting, the roller, once out of centre, is carried by centrifugal force outwards to the die, in which position it is underhung and not overhung, as with the Huntington mill.

As before, the force available for crushing is the centrifugal force arising in the gyration of the roller around the vertical axis. The number of such gyrations is determined by the speed at which the spindle is driven, in conjunction with the ratio of the roller and ring diameters. Ordinarily the roller is driven at about 300 r.p.m., at which rate the number of gyrations would be 200 per minute, and the crushing force about 4000 lb. This force is greater than with the Huntington mill, and the machine is able to crush larger material, say as large as $1\frac{1}{2}$ in. when not too hard.

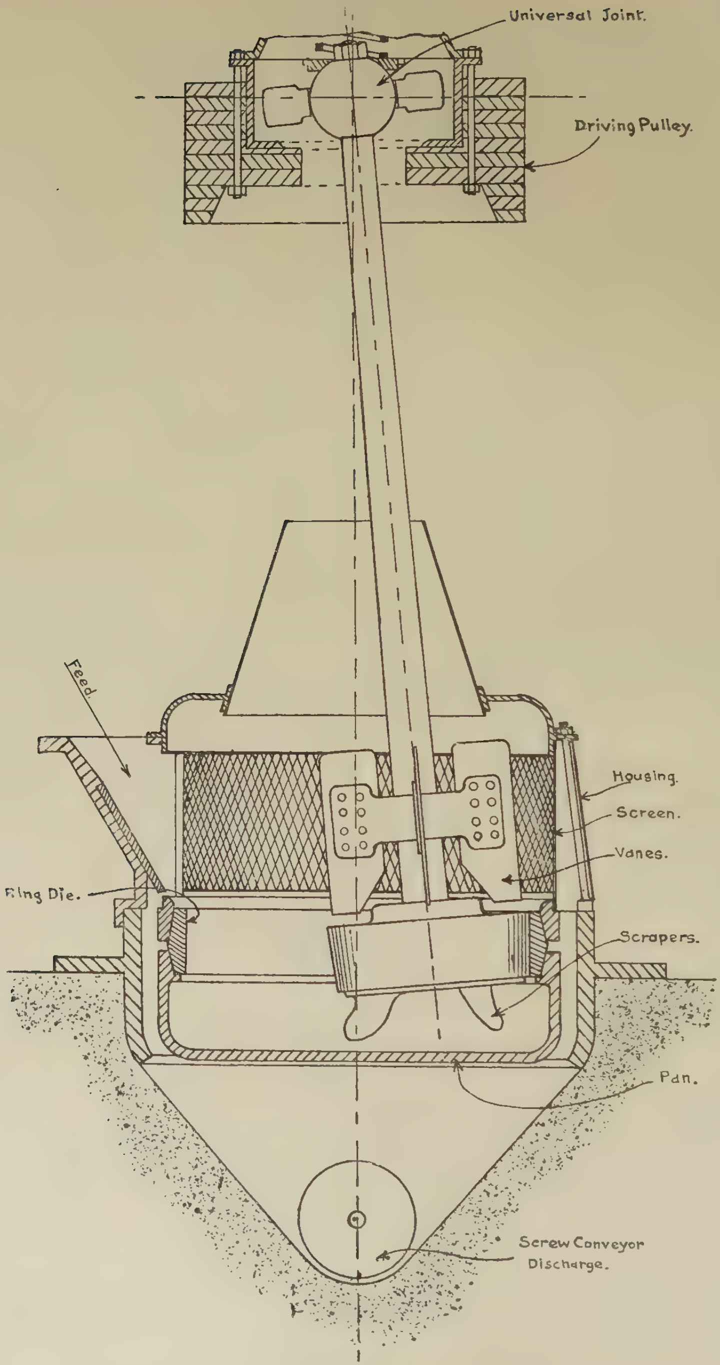


Fig. 56.

Griffin Mill.—General Drawing. The conical sheeting above the screen is to keep the dust from escaping; it is usually continued higher than shown (p. 85).

The feed is through a chute in one side which directs the ore into the crushing area. No water being used, the crushed material must be discharged otherwise. Scrapers attached to the underside of the roller keep the bottom of the pan clear, bringing the material up into the crushing area again, while vanes or blades attached above the roller blow the fine material outwards to a circumferential screen. Passing through this screen the fine material drops into a boot below the pan, from whence it is removed by a screw conveyor.

The screen aperture generally used is one corresponding to 30 mesh, and the ratio of size-reduction is consequently high, namely, about 100 : 1. Doing such work the machine, consuming about 25 h.p., will crush about 1 ton of ore per hour, equivalent to about 80 lb. per h.p. hour.

This machine, though not in general use with ore, was very successfully used at Kalgoorlie to crush telluride gold ores previous to roasting. In non-metalliferous mining, and particularly in crushing phosphate rock, it has a greater application ; it is also largely employed in making stone-dust for sprinkling in colliery ways. Upon this softer material its capacity is considerably greater and the wear of roller and die much less.

Edge-runners or Chilian Mills

In the typical Edge-runner or Chilian mill there are again two circular surfaces, one the cylindrical surface of a vertical roller and the other an annular die upon which that roller runs. In the earliest types this die was a flat circular disc around the edge of which the roller found its track ; hence the term Edge-runner (Fig. 57).

With these machines crushing takes place in this wise : the roller, in its run, rises upon material lying upon the die, crushing that material if its weight be sufficient. But that is not all. The surface of the roller being cylindrical, normal rolling would take it straight forward, whither, however, it cannot go, being constrained to take a circular path. The roller is in fact being continually pulled out of its straightforward path, with the result that in addition to crushing by pressure there is grinding by shearing. It may be assumed that the medial plane of the roller rolls truly, that is to say, in rolling it covers exactly the circular track allotted to it. The outside circle falling to the lot of the outer plane of the roller will, however, be greater, and to cover the greater length that outer plane must slide forward. Similarly, the inner circular track being smaller, the inside plane of the roller must be pulled back or it would overrun its course. In these twisting or pivotal motions there is grinding.

The relative parts played by pressure and grinding depend upon two factors, namely, the diameter of the die circle and the width of the roller face. It is obvious that the smaller the diameter the greater the part played by grinding, while, on the other hand, if the circle were infinitely great, that is, if the path were straight, there would be no grinding. With respect to width of face, though this factor is not so important, it is again obvious that the greater this width the more the grinding, while if the roller were a simple plane without width, there would be no grinding at all.

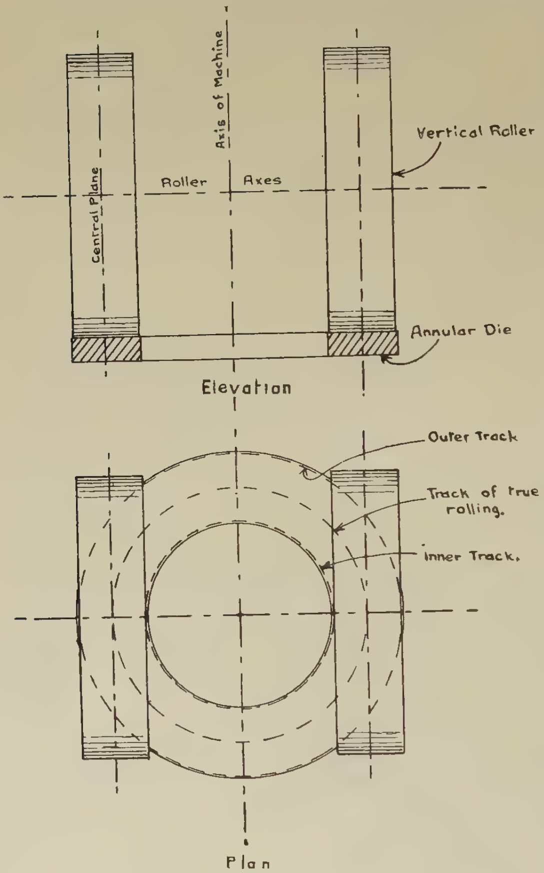


Fig. 57.

Chilian Mill.—Diagram of crushing surfaces (p. 87).

Two main types of these mills are recognized, namely, Slow-speed mills which crush almost entirely by pressure, and High-speed mills which crush largely also by grinding. Before describing these types, an Edge-runner may be mentioned which crushes entirely by pressure, no grinding action being developed. This machine is known as the Schranz mill. In it conical frustra revolve as rollers around the edge of a flat cone, the apices of roller and die cones coinciding. With such a disposition, differences in the circular track are precisely met by differences in the diameter of roller. Though this mill has no great application, it is interesting since it forms a link between rolls and Chilian mills, crushing like the former but having points of resemblance with the latter.

Slow-speed Chilian Mills (Fig. 58).—These mills are generally of large diameter but of relatively narrow roller-face, two factors minimizing grinding. On the other hand, the rollers are generally of sufficient weight to crush the ordinary product from a breaker; meeting such material in its path the roller mounts and falls again, a procedure which if rapidly repeated would give rise to serious vibration. These rollers consequently can only proceed at the comparatively slow speed of 8—16 r.p.m. around

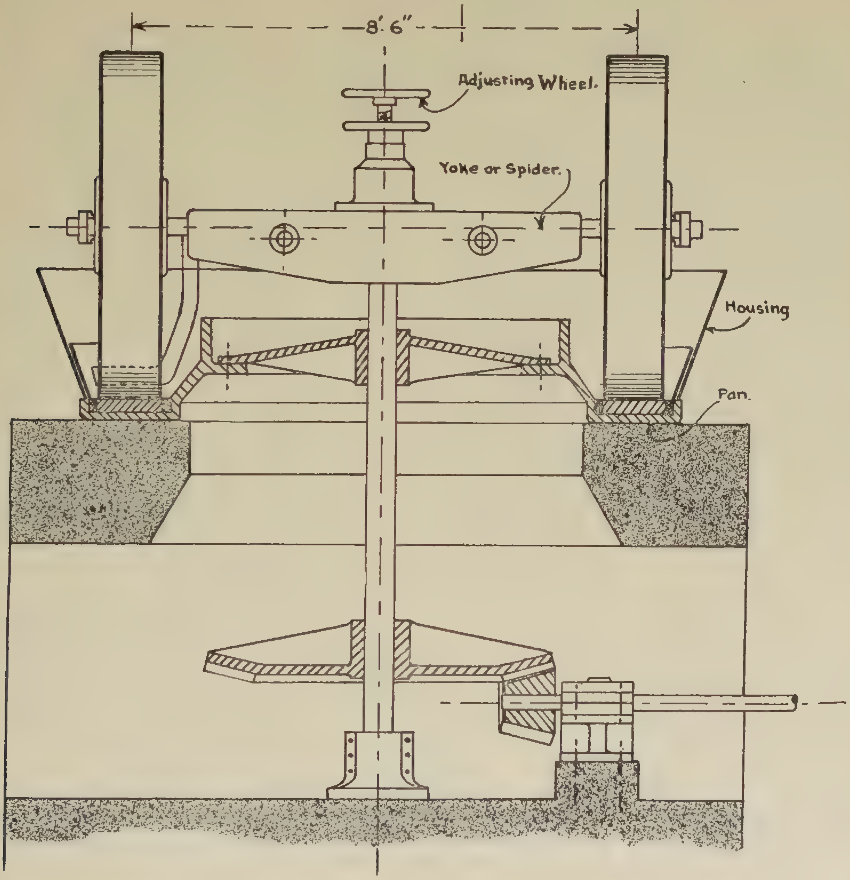


FIG. 58.

Slow-speed Chilian Mill (p. 88).—Bayldon, *Trans. I.M.M.* vol. xx., 1910, p. 125.

the circle, equivalent to a peripheral speed of 200—400 ft., the machine being 7—10 ft. diameter.

The weight of the roller is generally 5000—10,000 lb., the diameter 6 ft. or so, and the face about 9—15 in. There are, as a rule, two rollers, one at either end of the same diameter, though in special designs the number is increased to three or more. These rollers generally consist of a cast-iron core surrounded by a tire of suitable steel, this tire being sometimes as much as 9 in. deep and making one-third to one-half of the total weight when new, but wearing down to $1\frac{1}{2}$ in. and less, before being discarded. In particular designs, as with the Lane mill, the wheels themselves are light, the requisite pressure being obtained by weights in a tank above; in this mill also, there are often as many as six rollers.

The annular die lies flat in a pan-casting into which it is secured by wedges. Being about 3 in. thick and somewhat wider than the runner it is heavy, and for convenience made in sections. Upon this casting is reared the housing in which the screens are set, these generally leaning outwards. Around the track so constituted the rollers are moved from a central shaft which itself is generally driven through gearing from below, but at times from above. This central shaft passes up through the bottom of the casting to support by means of an adjusting screw, and to drive by means of a feather key, a yoke or spider, the vertical position of which can be adjusted to meet the wear of tire and die (Fig. 59). With two rollers, the arms of this yoke extend diametrically outwards from the shaft to contain the sleeves which serve as bearings for the roller spindles, these spindles while fixed in the roller being free to rotate in the sleeve. To prevent the roller from flying out from the centre in response to centrifugal force, the roller spindle has a collar at its inside end, this collar being held against the end of the sleeve, or in a special thrust-bearing. The sleeve itself has trunnions which are supported in the yoke in a manner permitting the roller to rise upon any unbreakable object without straining its connection with the driving apparatus or canting the diametrically opposite roller. Thus, not only is the centrifugal force tending to disrupt the machine held in check, but, the trunnions on the sleeve being below the axis of the roller spindle, the centrifugal force, in tending to bring them into the same horizontal plane, brings extra pressure to bear on the die, and the capacity of the machine is somewhat increased. In some particular designs this use of centrifugal force has been considerably extended by giving the roller spindles a relatively high inclination upwards and outwards.

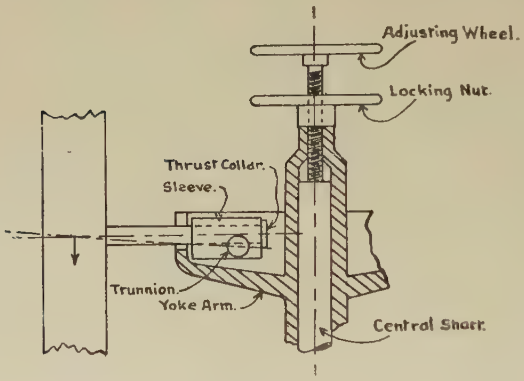


Fig. 59.

Slow-speed Chilian Mill.—Sectional drawing of driving mechanism and roller attachment (p. 90).

A further modification in spindle disposition is that known as the Mantney Offset, where the two spindles, though parallel, are not in line, but each a little ahead of the diameter which would otherwise connect them, the result being a drag upon the whole roller, the grinding effect being thereby increased (Fig. 60).

With regard to the two possible relations of the roller to its spindle, that where the roller is fast upon the spindle, the spindle turning in an independent bearing situated in the central mechanism, has been described. In other designs, however, the roller is loose upon its spindle, and the spindle does not turn but is held fast to the driving spider, of which indeed it may be part, a nut or thrust collar on the outside keeping the roller from flying outwards; the bearing being then inside the hub of the roller, must be protected from the sandy pulp continually streaming over the roller, by an oil cap covering the spindle end (Fig. 61).

These mills crush in water. The ore broken to about 1½ in. is fed at a point on the periphery, whereat water also is entered. By the continual passage of the rollers the ore becomes distributed, the water helping. Discharge as a rule takes place round a good part of the circumference, though sometimes the screen area is limited to a length directly opposite the feed entry. The amount of water used depends largely upon the extent of the screen area; if that area be large the amount of water required will be great and sometimes as much as 12 tons of water per ton of ore, whereas if it be limited to an opening at the front, the proportion of water may be as little as 2 tons. Naturally, the particular character of the ore also affects this question of the necessary water, since a clayey ore will require more water than one which is clean and brittle. In general it may be said that 5 tons of water are used per ton of ore.

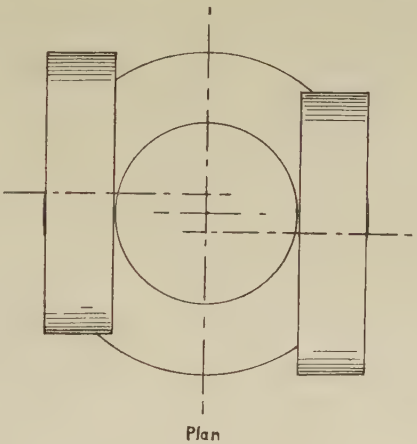


FIG. 60.

Slow-speed Chilian Mill.—Diagram showing rollers arranged with Mantney Offset (p. 91).

With regard to their product, these mills have their greatest application in crushing gold and silver ores to a fineness suitable for amalgamation or cyanidation, and it is usual for them to effect a complete reduction from about 1½ in. to the necessary fineness, a screen of about 30 mesh being common. Size of screen aperture is, however, not the only factor determin-

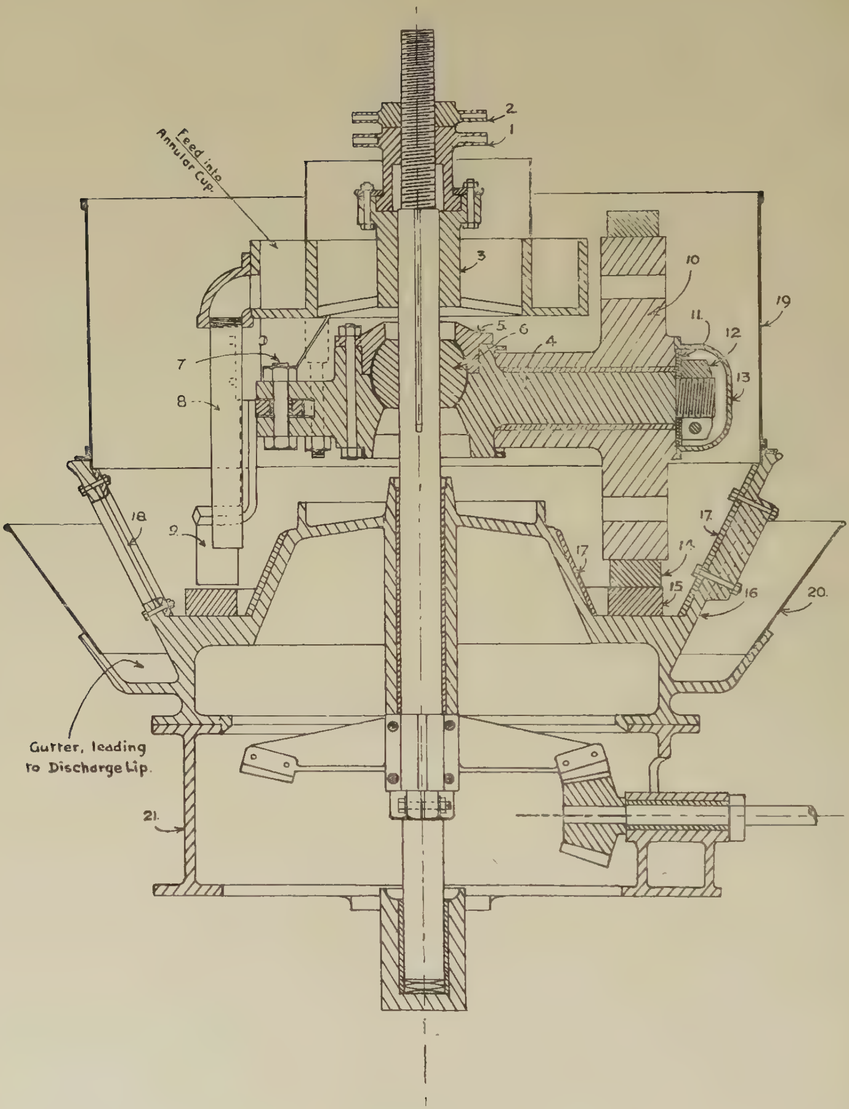


Fig. 61.

1. Adjusting Nut.
2. Locking Nut.
3. Yoke.
4. Spider.
5. Spider Cap.
6. Spider Ball.
7. Driving Links (3).
8. Feed Pipes (3).
9. Scrapers (3).
10. Rollers (3)
11. Thrust Plates and Washers.
12. Thrust Nuts.
13. Oil Covers.
14. Roller Tire.
15. Die.
16. Pan or Mortar.
17. Liners.
18. Screens (6).
19. Steel Housing.
20. Splash Guard.
21. Base Frame.

High-speed Chilian Mill (Evans Waddell).—This mill has three rollers which, being loose on their spindles, are held by thrust washers and nuts on the outside. The spindles are the arms of a spider; they meet at the centre upon a spider ball, a design which permits play as the rollers rise upon larger pieces. This spider is driven by links from a yoke above, in which yoke the annular cup into which the feed is entered, is arranged. Spider, cup, and yoke slide along a feather key upon the driving shaft, their position upon that shaft and consequently the horizontal plane of the rollers being adjustable by nuts (pp. 91, 93).

ing the fineness to which the material is crushed. The ore being crushed in water is carried up by that water to be discharged, and the height so lifted, known as the "height of discharge," largely enters the question of the ultimate size of the crushed product. Were that height great only the very finest material would be discharged, in fact, these mills are sometimes run without screens, the required fineness being obtained by a high discharge. Ordinarily, it varies from 6 in. to 15 in., these figures at the same time representing approximately the depth of water above the die. There is little point about making it too shallow, since the lower portion of the screens would then become blocked by sand; moreover, the commotion in the water is always sufficient to keep the fine material suspended to a height well above the die.

Considering the great reduction in size these mills effect, they have a relatively great capacity, crushing 20—30 tons per day or 150—200 lb. per h.p. hour. They are simple machines, permitting low operating-cost and costing little for repair. They have a high percentage of working time and work with little vibration or noise. Against these several advantages is the solid disadvantage of their great weight, which makes them costly and unsuitable where transport is difficult; and the further disadvantage of the large floor-space they occupy. They have, however, been much used in Russia and Mexico for crushing gold and silver ores, their slow speed and fine-crushing permitting efficient amalgamation, while producing material in a good condition for cyanidation. For crushing material preparatory to concentration they are not suitable, nor are they used to advantage on small material.

The cost of crushing by these slow-speed mills is generally about 12d. per ton. Tires and die last about a year, the total consumption of steel being 1—1·5 lb. per ton. Repairs and renewals are few. Power consumption also is low, only a proportional amount being consumed when the machine though running is not fully fed; moreover, experience shows the capacity is not greatly affected as the tire and die wear.

High-speed Chilian Mills (Fig. 61).—These mills are in most respects similar to those just described. In diameter of track and weight of roller, however, they are considerably smaller, the diameter being generally 5—6 ft., and the weight of roller about 3000 lb. To offset their smaller dimensions these mills are run at a higher speed, 30—40 r.p.m., and normally have three rollers in the place of two. In consequence, and also because of the smaller size of material fed and a lower discharge, they have a high capacity, 60—100 tons per day; consuming 40—50 h.p., they crush about 180 lb. per h.p. hour.

The material fed is that which has already been crushed to $\frac{1}{4}$—$\frac{3}{8}$ in. It is delivered with water into an annular cup revolving with the central spider, from whence, through pipes, it drops on to the track between the rollers, an even distribution thereby resulting. The crushed product is delivered through screens—generally of 20—30 mesh—set at a low height of discharge, so that the particles smaller than the screen aperture get quickly away.

Though with such small material the necessity to give each runner freedom to rise over large pieces is less, it nevertheless exists, and provision for such freedom is part of the design. In addition, as with the slower mills, the whole driving mechanism may slip downward on the central driving shaft as the tire and die become worn, and be raised again by an adjusting screw to suit new wearing parts.

It is usual with these mills to have the roller loose upon the spindle; the spindle then is solid with the driving spider, extending right through the roller to hold this latter against centrifugal force by a thrust collar or nut on the outside. At such a high speed this appears to be the simplest way of meeting the thrust, a special oil cup over the collar permitting efficient lubrication. The roller then turns on the spindle, a sufficient length of bearing being obtained by extending the roller hub towards the central axis.

With such a high speed and three rollers in the place of two, it is usual to find that the die wears somewhat quicker than the tires. The life of both, however, is short and generally about three months, by which time they have lost about three-quarters of their original weight. Including the weight discarded this wear is about 0·5—1 lb. per ton crushed, the precise amount depending upon the character of the ore and the class of steel.

These machines have their greatest application as intermediate crushers to give a granular product. Exceptionally, with relatively soft ore they have been used to crush coarser material to the necessary fineness for amalgamation and cyanidation, but their particular usefulness has been in the re-crushing of middling and tailing for further concentration. Though the amount of size-reduction they effect is small compared with that of the slower mills, their operating cost is much the same, namely, about 12d. per ton crushed. Their higher speed increases the amount of repair and makes them unsuitable to reduce material of larger size.

GRINDING PANS

When introducing edge-runners it was stated that, apart from comminution by pressure, comminution also took place by shearing, this latter varying in amount with the twisting or pivotal action developed. It was further explained that this pivotal action increased with the smaller diameter of the mill and with the greater width of the roller-face. In the Grinding Pan, which comes next in the sequence of crushing appliances, the width of the wearing faces is great and the diameter of the pan relatively small; moreover, the two faces no longer have a rolling but a rubbing contact. These factors increase grinding by shearing at the expense of crushing by pressure (Fig. 62).

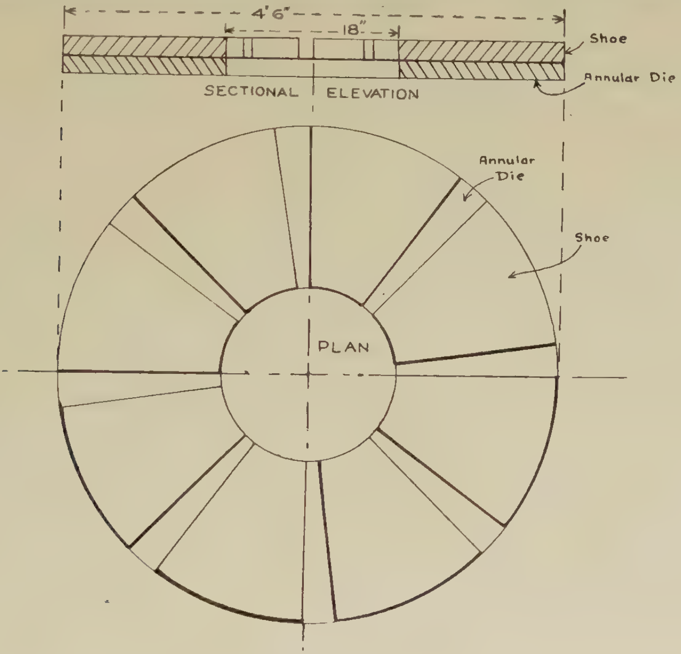


FIG. 62.

Grinding Pan.—Diagram of Crushing Surfaces. The die sections make a complete ring and occupy roughly the outer two-thirds of the pan diameter, or, say, about four-fifths of the pan bottom. The separate shoes sweeping around the same annular space, generally alternate with unfilled spaces, though sometimes they too completely fill the ring (pp. 95, 99).

In these pans there is again the same fixed horizontal annular track or die as with the Chilian mills, though now its width is considerably greater. Upon that die, instead of rollers, shoes held in a horizontal annular ring, known as the muller ring, revolve, the sand particles coming between these two grinding surfaces being held by the weight of the revolving shoe and sheared by its movement. Obviously this means of comminution is only suited to the further reduction of material already in the condition of sand, and accordingly the pan is used as a fine grinder, to grind, for instance, the coarser portion of stamp-crushed material to a better condition for amalgamation or cyanidation. These pans were indeed formerly known as amalgamating pans, use being made of them to grind, and while grinding to amalgamate, charges of gold or silver ore. They are, however, now also employed as purely grinding machines, particularly in Australia upon gold ores, and in Cornwall and Bolivia upon tin ores.

The pan proper is usually a cast-iron vessel 5 ft. in diameter and 2—3 ft. deep (Fig. 63), having in the bottom a ring die. This ring die is divided in sections, each with a lug fitting into a recess so fashioned in the pan bottom that any tendency to move with the revolving shoes only results in further tightening. In the centre the bottom rises as a steep hollow cone to provide passage for the central spindle and a seat for the brass sleeve forming the actual bearing for that spindle. On the top of this spindle the adjusting mechanism determining the exact height of the shoes in relation to the die, is set. This mechanism consists of a screw having at its end a hand wheel, and a saddle through which the screw passes; this screw bears on the top of the spindle, and the saddle is raised or lowered by turning the screw; above the saddle and also on the screw is a wheel nut to secure the saddle after adjustment and therefore known as the locking wheel. Hanging from the saddle comes the yoke, a piece which in its upper portion embraces the central spindle closely, and in its lower portion expands to cover the bearing through which that spindle passes. This yoke being bolted to the saddle rises and falls with it, sliding on a feather key fitted on the spindle. To the bottom of this yoke is fixed a four-armed spider which supports the muller ring, to the under side of which the shoes are attached. These shoes are sometimes of a shape and number completely to occupy the circle, whereas at other times unoccupied spaces intervene; each separate shoe weighs generally about 100 lb. and is about 2½ in. deep. They are held in the muller ring either in the same way as the dies in the pan bottom, or by loops which projecting upwards above the muller are caught by wedges (Fig. 67). Their weight, inclusive of the muller, is the measure of the pressure available for crushing; divided over the relatively great area that pressure is small, and comminution by pressure is subordinate. The more important duty of the shoe is so to grip the larger particles that upon revolution they become sheared. No pressure can be exerted by the adjusting screw, the revolving masses bringing only their own weight to bear. The central spindle is driven by bevel gearing from below, motion being communicated to the yoke by means of the feather key.

Though all grinding pans are very similar there are differences which justify the separate description of some types.

Wheeler Pan (Fig. 63).—This was originally an amalgamating pan in which ore was treated in charges. When applied to continuous grinding, the ore, in the condition of a water-borne pulp having a consistency of 2—5 of water to 1 of ore, is brought by a launder into the body of the pan where the muller is rotating at about 50 r.p.m. At this speed the coarse particles seek the periphery, and, to direct them back again to the grinding zone, deflecting wings are bolted to the sides. Eventual discharge takes place by peripheral overflow, the height of discharge, or, in other words, the depth of the pan, governing the fineness of the product overflowing. Discharge may be either all round or at one point. In the latter case the overflow lip leads into a pyramidal box, at the bottom of which a hole is

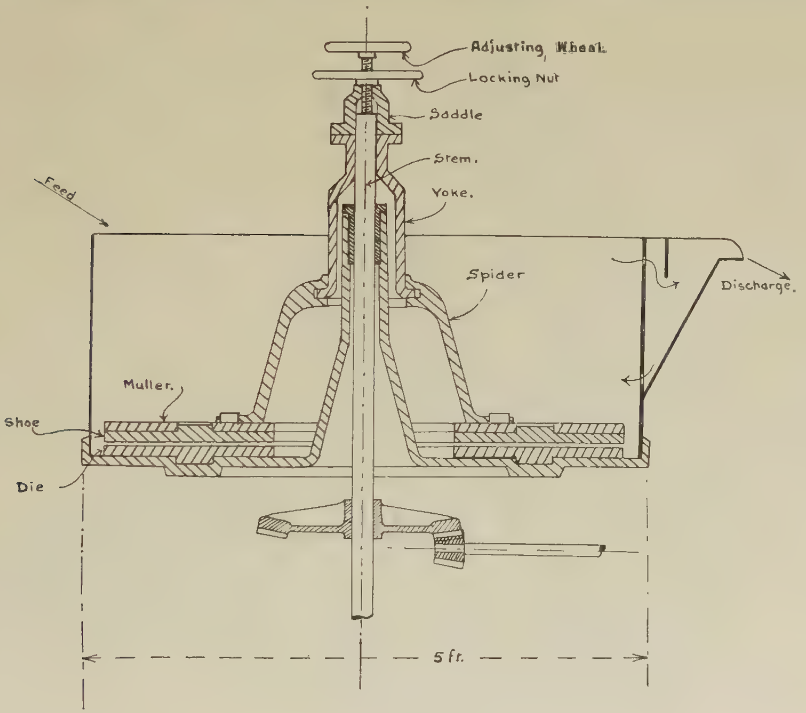


Fig. 63.

Grinding Pan (general type).—Wheeler Pan (pp. 96, 97).

provided through which any coarse particles prematurely carried over are returned. From this classifying box the properly finished material has no difficulty in escaping. The sides of the pan are generally cast with the base, though sometimes they are made of wooden staves having proper seating in a basal casting. Generally also they are vertical, though in the original pan they were slightly splayed outwards.

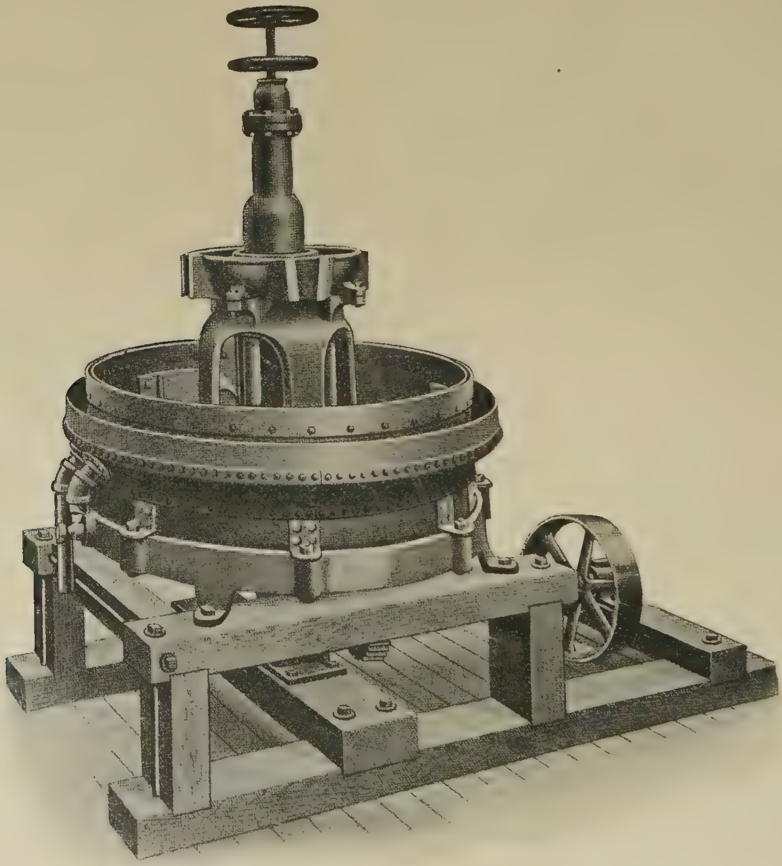


Fig. 64.

Grinding Pan (Colorado Ironworks).—General Appearance. This illustration shows annular feed cup and vertical pipes depending therefrom; spider arms; deflecting vanes on interior surface; discharge over the entire rim into a peripheral gutter or launder; dovetailed seatings on the outside of the feed cup for weights to compensate for wear of shoes (p. 99).

In this pan the shoes do not usually fill the complete ring, but intervening space gives opportunity for the sand to settle in the way of the next oncoming shoe. On the other hand, the dies occupy the complete circle except a narrow space be left between each two dies to promote circulation of the sand. The wear of shoe and die is great, amounting generally to well over 1 lb. per ton, and sometimes to as much as 4 lb., depending upon the character of the ore and the class of iron or steel. As this wear proceeds, the shoes are kept in proper contact with the die by lowering the muller. The loss of shoe weight cannot, however, so easily be made good. Starting with a thickness of about 2½ in. they are worn to about ½ in. before being discarded, and the difference in weight between new and old shoes is considerable. This difference can be made good by compensating weights upon the muller ring, though this practice has not become general (Fig. 64). In the grinding pan, using pressure only to a secondary extent, no proportional drop in capacity takes place as the shoes wear. A set of shoes and dies generally lasts 3—6 months.

The power consumed is about 6 h.p. and the capacity some 20 tons per day, equivalent to 275 lb. per h.p. hour. The size-reduction, however, is not high, but generally about 5. If set to crush finer, the grinding surfaces come in contact, the capacity diminishes, and wear increases. They cannot in consequence be satisfactorily employed to effect the finest comminution.

Modifications of the Wheeler pan have largely consisted in altering the relative positions of feed and discharge. In the type described the feed was into the body of the machine, or to one side, while discharge was over a high-level lip at an opposite point on the periphery. In spite of its height some coarse material is thrown over this lip by centrifugal force, this force throwing the coarse to the periphery and keeping the fine at the centre. Central discharges have therefore been arranged in some machines, as in the Freeman and Holman pans, with some advantage in the fineness of the product. The Freeman pan is the Wheeler pan arranged with discharge through a pipe which, passing through the side of the pan about half-way up the side, extends well into the pan interior, where its orifice is turned to meet the swirling pulp.

Positive Pan (Fig. 65).—Working on other lines, and following the idea that the material should be made to pass between the grinding surfaces, other pans have been designed with a central feed and a low-level peripheral discharge. In these pans the material is fed into a central sheathing, the only escape from which is between a completely filled shoe-ring and the die. Discharge then is facilitated by centrifugal force, and the degree of crushing effected depends upon the relative velocities outward of the pulp and circumferential of the shoe. In the outward movement of the pulp the consistency of this latter plays a considerable part; if the pulp be too thin it moves too quickly and reduction is incomplete; a consistency of 2 of water to 1 of solids has been found good practice. Escaping from between the grinding surfaces, crushing, as far as the particular pan is concerned, is finished, there is no possibility of return; accordingly, no advantage results from great height of discharge, but on the contrary, a useless consumption of power in giving motion to a great volume of pulp. These pans therefore run with a low discharge, the

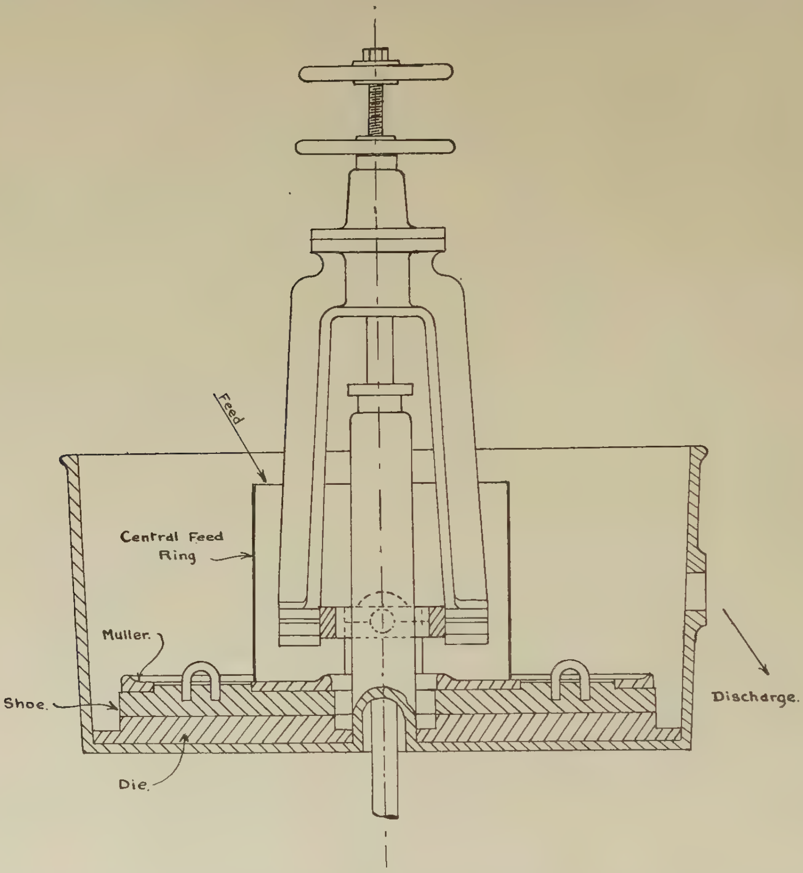


FIG. 65.

Positive Pan.—The type illustrated is that developed in Western Australia. The muller hangs from two trunnions provided at the bottom of the yoke. The shoes are attached to this muller by loop and wooden wedge. From this muller rises also the inner partition into which the feed is entered. The discharge is through a hole about half-way up the pan side. A pan embodying similar ideas was employed by Söhnlein to grind tin ores in Bolivia (p. 99).

capacity is relatively high and the power consumption correspondingly low. By thickening the feed still further it has been found possible to reduce sand to the size of slime.

Holman Pan (Fig. 66).—This pan, which is largely used in Cornwall

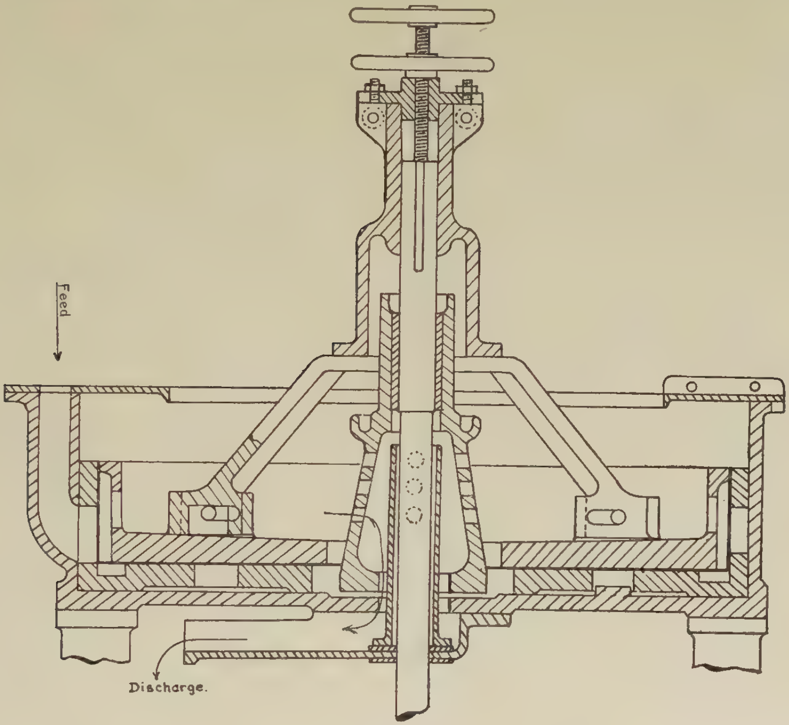


Fig. 66

Holman Pan.—Though in appearance much similar to the general type, this pan is seen to be shallower. The feed is entered at a point on the periphery from whence it makes its way all over the pan through flutings and recesses. The shoes bear on the bottom, but are also thrown against the sides, their slotted connection with the spider arms permitting this outward movement. Discharge eventually takes place through holes in the central cone, passing thence out at the bottom, the lower part of the driving spindle being protected from this discharge by a special sleeve (pp. 99, 101).

for regrinding middling and other products, has, as before stated, a central discharge; discharge takes place through holes in the central cone, the feed being introduced at the periphery. This pan differs further from those of the Wheeler type in that the shoes grind not only on the bottom but to some extent also on the sides, against which they are thrown by centrifugal force. Working thus, the shoes are of special shape, and the pan is protected by side liners. Further, being shallow, the pan is largely covered on top to prevent pulp being thrown over the sides.

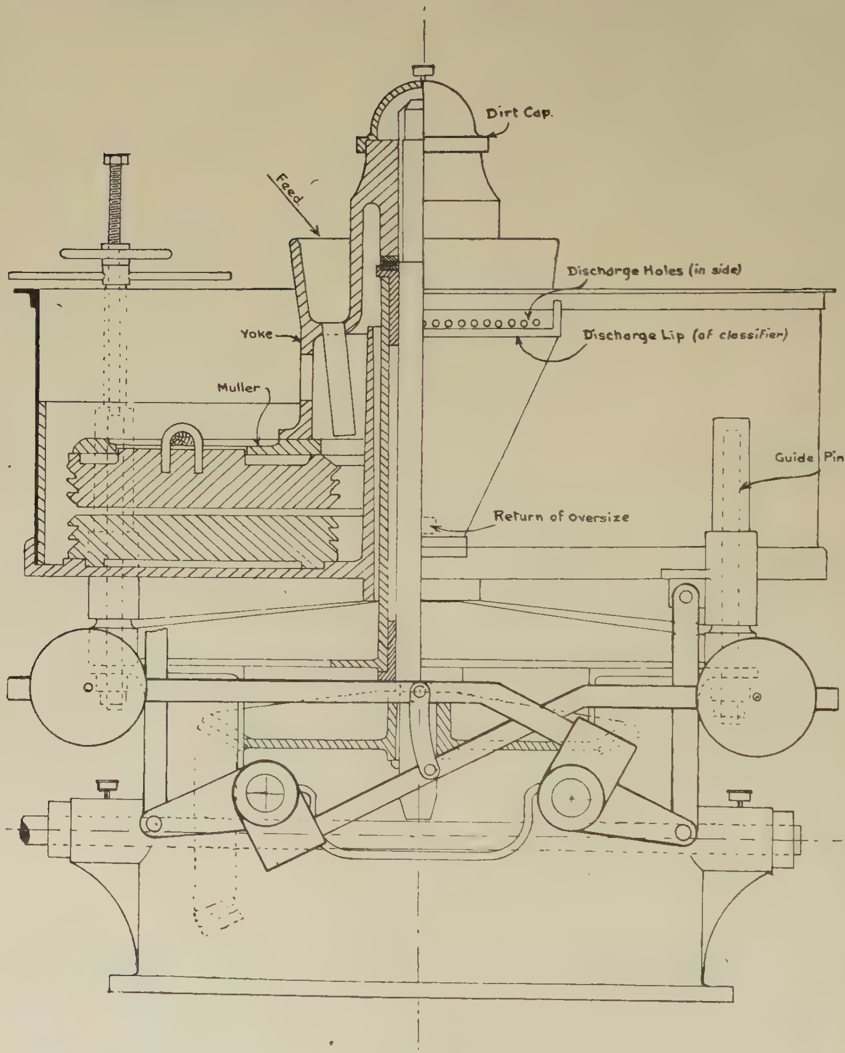


Fig. 67.

Cobbe Pan.—The feed is seen to be central and the discharge peripheral. The feed pipes take the sand so far down into the interior of the pan that the only escape is between the grinding faces. This escape is not easy since the shoes make practically a completely filled circle. The regulation of the proper aperture between the crushing faces is by means of adjusting and locking nuts on a screw outside the pan (pp. 96, 103).

Cobbe Pan (Fig. 67).—This pan is similar in most respects to the Wheeler pan, from which it differs, however, in the endeavour it makes to eliminate the varying weight of the shoe ring as wear proceeds. This it does by holding that ring steady under its own weight, the die ring being forced up to it from under, by an arrangement of four weighted levers not sufficiently powerful to lift the shoe ring. The pan bottom and the die ring have for this purpose freedom to move 6 inches upwards from the position they occupy when shoe and die are new, while the diminishing weight of the die as it wears is compensated by removing weights from the levers. Against the uniformity of weight and the greater capacity thereby resulting must be set the greater complication in design.

Résumé.—Pans are efficient fine grinders but compared with tube-mills—now to be described—they cannot satisfactorily accomplish the finest comminution. The operating cost, though the consumption of power is low, is inclined to be high owing to the smallness of the unit, and is about 1s. 6d. per ton. To obtain the economy of a larger unit some pans have been made 8 ft. in diameter, but all points considered these have not shown themselves superior to those of standard size. Pans are now only used where first cost is an important consideration. They are cheap, five pans not costing half as much as one large tube-mill, and consequently in demand for small installations, more particularly since they are simple both to install and to operate. They have been largely used for grinding and amalgamation in America and Australia; for the simple grinding of gold ore previous to cyanidation in Australia; for the fine grinding of the lead-silver ore at Broken Hill previously to concentration; and for the fine grinding of tin ore in Cornwall.

CYLINDER MILLS

(*Including Tube-Mills, Ball-Mills, etc.*)

GENERAL

In cylinder mills comminution is effected by the rolling and falling of balls, pebbles, rods, etc., within a revolving cylinder, into which the ore is fed at one end and discharged at the other. These balls, etc., are the crushing bodies, and though exceptionally a single body may weigh as much as 100 lb., the normal maximum weight when of steel is 40—50 lb., and when of flint about 2 lb. Accordingly, and even though with a heavy charge of balls there will be pressure upon the lower layers, pressure largely disappears, shearing and impact taking its place.

Speed and Diameter.—With a proper charge, as the cylinder starts from rest, friction with the side and between the balls takes the ball charge round, till the slope of its surface exceeds the angle of repose, when the balls above that angle break away and roll down the slope, grinding and crushing as they go. A normal angle of repose in water would be about 30° (Fig. 68).

As the speed increases centrifugal force holds the balls tighter to the side carrying them higher, till gravity, setting itself now against centrifugal force rather than against friction within the charge, overcomes that force, and the balls, leaving the circle, take flight along a parabolic path, the resultant of their acquired velocity and gravity. Falling finally to the side again they crush by impact, and then struggle against the rotation till, their inertia overcome, they start the cycle afresh.

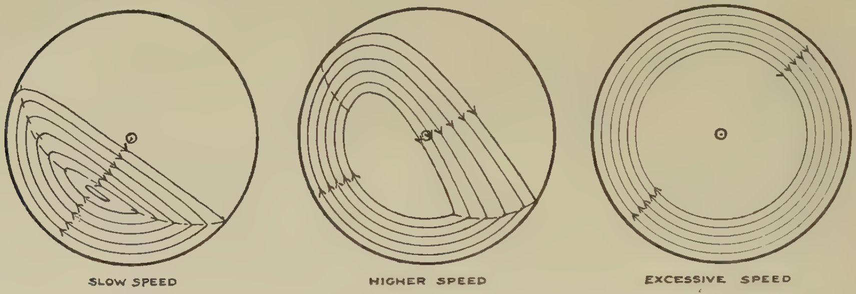


Fig. 68.

Cylinder Mill.—Diagrams illustrating Ball Movement. The first figure illustrates the movement at that slow speed of revolution which lifts the balls till, the angle of repose being passed, the upper ones of each layer break away and roll down the slope, crushing resulting chiefly from grinding. The two dotted lines mark, respectively, the breakaway from the circular path and its resumption.

The second illustrates the parabolic flight at that speed when the balls of the outer layer, and those only, in dropping back into position to resume the cycle, strike the cylinder side; crushing is now by impact chiefly. Again the two dotted lines mark the breakaway from the circular path and its resumption; it will be noted that the extent of circular path is now greater. At this speed about half the balls are in the air.

In the third figure the position resulting from excessive speed is illustrated. All the balls now hug the periphery; there is neither breakaway nor resumption; the extent of the circular path is infinite.

In all three figures the ball charge is the same; it will be noticed how the relatively quiescent central area increases with speed (p. 104).

At still greater speed gravity finally becomes powerless against centrifugal force, and the balls no longer leave the circular path nor alter position among themselves; in such circumstance no comminution is possible.

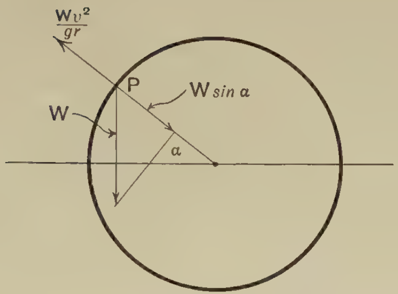


FIG. 69.

Cylinder Mills.—Diagram of Forces acting on a Ball within a Cylinder Mill. If a ball of weight W hugs any point on the inside periphery of a revolving cylinder, it is held in that position by centrifugal force acting radially from the centre against the component of gravity resolved along the particular radius, the relation of these two forces being such that centrifugal force is greater than the gravity component. Taking the angle made by the particular radius with the horizontal to be α, this relation is expressed by the inequality

$$\frac{Wv^2}{gr} > W \sin \alpha.$$

That this hugging of the periphery should be continuous all round the circle, centrifugal force must be greater than the maximum value of W sin α, that maximum being reached when α is 90° and $\sin \alpha = 1$; continuous hugging, accordingly, obtains so long as

$$\frac{Wv^2}{gr} > W, \text{ or } v^2 > gr.$$

But v, the peripheral velocity, is equal to $n\pi d$, where n is the speed of revolution, and d the diameter in place of the radius. Using this value of v, the formula giving the limiting speed beyond which the ball would cling continuously to the periphery, that is the 'critical speed,' is thus derived:

$$n^2\pi^2d^2 = gr. \text{ whence}$$

$$n = \sqrt{\frac{g}{2\pi 2}} \times \frac{1}{\sqrt{d}} = \frac{1{\cdot}277}{\sqrt{d \text{ feet}}} = \frac{0{\cdot}705}{\sqrt{d \text{ metres}}},$$

or, with n as the speed per minute (instead of per second)

$$n = \frac{76{\cdot}6}{\sqrt{d \text{ feet}}} = \frac{42{\cdot}3}{\sqrt{d \text{ metres}}}.$$

In practice, the running speed is about 75 per cent of the critical speed proper to the clear internal diameter of the mill, and no ball continuously hugs the periphery. But the balls do not all leave their circular path along the same inclined radius; those of each layer respectively break away when

$$\frac{Wv^2}{gr} = W \sin \alpha, \text{ or when}$$

$$\sin \alpha = \frac{v^2}{gr} = \frac{n^2 4\pi^2}{g} r = Cr.$$

That is to say, with increasing radius the balls, irrespective of their weight, climb farther up the circular path before they break away (p. 104).

This last position must not be reached. For each layer of balls there is a "critical speed" of revolution at and beyond which it obtains, a speed calculable by equating gravity and centrifugal force (Fig. 69).

Thus,
$$\frac{Wv^2}{gr} = W\text{ ; or } v^2 = gr,$$

v being the peripheral velocity,

whence
$$n = \frac{76{\cdot}6}{\sqrt{d \text{ feet}}} \text{ or } \frac{42{\cdot}3}{\sqrt{d \text{ metres}}},$$

where n is the number of revolutions per minute, and d the diameter of the circular path.

It is seen that the smaller the diameter of any circular path the higher its critical speed—its peripheral speed, on the other hand, is lower. Accordingly, where there were several layers of balls the first to be held fast would be those at the outside, this taking place while those of the inner layers were yet in free flight. The critical speed of the mill must accordingly have reference to the outside layer, and mills are usually run at about 75 per cent of the critical speed pertaining to their clear internal diameter —somewhat slower when crushing fine material or using heavy crushing bodies. In practice the revolutions per minute vary from 20 with mills of, say, 8 ft. diameter, to 33 with mills of, say, $3\frac{1}{2}$ ft.

At such speeds the outer balls fly, each in separate flight, to fall upon the side and then roll into place to start the cycle again. So also with each layer in turn; there is no general mix-up of the balls, but each keeps roughly to its layer, the larger to the outside and the smaller to the inside. Nearer the centre, however, flight becomes less certain, suffering interference wherefrom rubbing and rolling result. Doubtless also there is some slip of the charge as a whole, and some adjustments of the balls among themselves, both movements being accompanied by grinding. Comminution in cylinder mills therefore takes place partly by grinding and partly by impact-crushing. Within limits, the slower the speed and the greater the volume of the ball charge, the greater the relative part played by grinding, while with a smaller charge and a higher speed relatively more crushing is done by impact.

Concerning mill diameter, the range of which was given above, all other things being equal, large material will require relatively large diameter in order to develop a sufficient force of impact. But large diameter connotes greater peripheral speed before flight begins, more power being thereby consumed. Accordingly, when crushing large material the individual weight of the ball should be increased, rather than that the greater impact necessary should be obtained wholly by increasing the diameter.

Volume of Charge.—The volume occupied by the crushing bodies within the mill is an important factor. If the charge be too light, normal friction with the side will be insufficient to raise the balls even to the angle of repose, the balls will slide or roll back and no adequate work will be done. If, on the other hand, the mill be too full, there will be insufficient room for flight and the work again will suffer. Where, therefore, crushing by impact is desired the mill should be less than half full; but where reliance is placed upon grinding, the mill is probably most efficient when more than half full. About seven-sixteenths of the internal volume is a common charge.

Crushing Bodies.—Under given conditions of mill speed, diameter, and charge, the size of ball is determined by the maximum size of the feed: large material requires a heavy ball. Balls much below full size are of little use in a mill, since they are forced to the centre where their small weight is rendered still less effective by diminished movement. On the other hand, to have balls too large would be to reduce unnecessarily the number of contacts in a given volume of charge, and the number of blows in a given time.

In practice, balls are of flint, steel, or an iron-manganese composition. Flint balls are well-rounded pebbles of about 4 in. maximum and 2½ in. minimum diameter. Steel balls are generally of forged or chrome steel with 7 in. maximum and 1¼ in. minimum diameter. A mill charged with steel balls, supposing it were capable of sustaining the weight of such a charge, would have from one-and-a-half times to twice the capacity of a mill of the same size charged with flint pebbles suitable to the same class of work.

Latterly, in a particular mill, the Marathon or Rod mill, steel rods, normally of 2 in. maximum diameter and the length of the mill, have been used as crushing media. In spite of the great bulk of these bodies, the opportunities for crushing are not diminished but rather increased, lines of contact taking the place of points. A greater weight of charge can also be contained in the same volume, and a smaller mill suffices.

Mill Length.—The considerations so far have been centred around but one element of the length, a single cross section. Considering now the whole cylinder, there would be the same distribution and behaviour of the balls throughout the length, and in consequence the same capability in respect to crushing. But the material undergoing comminution becomes successively finer, and the capability determined for the entry might well be wasteful at the discharge. Cylinder mills should accordingly be short, or if they be long there may be no extreme

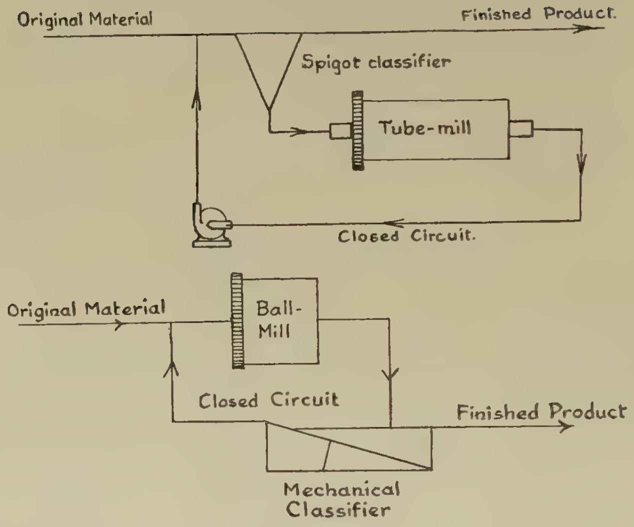


Fig. 70.

Cylinder Mills.—Diagrams illustrating mills working in closed circuit (pp. 109, 119, 123). In closed circuit the relation of the weight passing through the mill to that of the original feed may be determined from sizing analyses at different points around, and in the circuit. This sizing analysis consists of a division into oversize and undersize by a laboratory screen with apertures roughly equal to the maximum size of the finished product, through which screen accordingly the great bulk of that product passes; where in the circuit an actual screening appliance is employed, as, for instance, with rolls, the sizing analyses would be conducted with a screen of the same aperture.

Thus; In the tube-mill circuit illustrated:

Let A = weight of original material, and consequently of finished product.
B = weight of material passing through the tube-mill, and consequently of the material in the closed circuit.
w = percentage of undersize in original material.
z = percentage of undersize in finished product.
x = percentage of undersize in classifier discharge (coarse).
y = percentage of undersize in mill discharge.

Then, the weight of undersize entering the classifier being the same as that leaving it,

$$Aw + By = Az + Bx, \text{ or } B = A\left(\frac{z - w}{y - x}\right).$$

In the ball-mill circuit the original material as illustrated is generally the unscreened dry product from a breaker, such being fed direct to the mill without any endeavour to separate the undersize.

Let A = weight of original material, and consequently of finished product.
C = weight of material in the closed circuit.
B = weight of material passing through the mill = A + C.
w = percentage of undersize in original material.
z = percentage of undersize in finished product.
x = percentage of undersize in classifier discharge (coarse).

Let y = percentage of undersize in mill discharge.

Then,
$$(A+C)y = Az + Cx$$
$$C = A\left(\frac{z-y}{y-x}\right)$$
$$B = A + C$$
$$\therefore B = A + A\left(\frac{z-y}{y-x}\right)$$
$$= A\left(1 + \frac{z-y}{y-x}\right).$$

difference in size between the material entering and that leaving. Where the material sent to such mills is very fine already and only requires to be reduced to slime, this condition is largely fulfilled, long length is then a security against the escape of a particle uncrushed; but where the feed is coarse a relatively short mill is imperative. In either case, still better conditions are obtained by hurrying the material through the mill, separating that portion crushed fine enough, and returning the oversize to the mill again. In that way much of the material at the discharge end is returned to the entry, and undue difference in average size between the materials at the two ends of the mill is avoided. This separation of the oversize is generally performed by a water-sizing apparatus, known as a classifier, working in "closed circuit" with the mill, from which circuit only completely crushed material escapes (Figs. 77, 81). The amount of returned oversize is then generally a multiple of the original material (Fig. 70).

If instead of thus working in closed circuit there be no return of oversize the circuit is said to be open. From the foregoing, a mill so working should be shorter and more fully fed with original material. With too short a length, however, the number of particles escaping uncrushed would be unreasonably great; using balls or pebbles there is no enforced passage of each particle into the crushing space around the points of contact, and conceivably a given particle might pass from end to end and not be hit. Accordingly, other than simplicity there is not much to recommend the open circuit. Lengths vary from 4 ft. to 24 ft.

Rate of Feed.—From what has just been said, the rate of feed, original and returned material together, will depend largely upon the size-reduction to be accomplished. With coarse material and short mills it should be such that, at a dilution of one of water to one of ore, the pulp in the mill should be renewed every 1–2 minutes; with fine material and a long mill, on the other hand, this renewal of the pulp should take about 20 minutes. Knowing that the space occupied by the pulp is that between the balls and up to the pulp level—this space being generally

12—20 per cent of the interior volume of the mill—the proper rate of feed for any particular mill may readily be computed.

Dilution of Feed.—Water, when used in a cylinder mill, serves by assisting feeding, progression, and discharge. In respect to feeding, this assistance is felt particularly when the material fed is fine enough to be water-borne, but even with large material some water is necessary to flush the intake. Assistance in progression comes because progress takes place largely by the displacement of the crushed material by that newly arriving, such displacement being promoted by water. Finally, whether discharge takes place purely by overflow, by lift, or through apertures, it is always favoured by water. Exceptionally, as where roasting follows, or as in the manufacture of cement, a mill may be worked dry, that is, without water; the progress of the material from feed to discharge then is slower and the mill capacity less.

Though water renders these assistances its presence has disadvantages. It forms a pool in which the crushing bodies lose weight and energy; it also cleans their surfaces and that of the mill lining, to their greater wear; and at the same time it permits the particles requiring grinding to settle out, leaving the upper balls to beat themselves, wasting energy. To lower the pool, if the balance of advantage be that way, the height of discharge must be lowered, either actually, by enlarging the discharge opening, or virtually, by lifting the material to the discharge. To keep wear at a minimum the pulp should be thick enough to constitute a protective covering to balls and lining, when at the same time the wasteful beating together of the balls is avoided. Moreover, with a thick consistency the pulp to some extent is dragged upward with the balls and the pool is lowered (Fig. 78).

The amount of water is therefore kept to the minimum required for proper progress through the mill and eventual convenient discharge. Experience indicates that when crushing to produce fine sand the weight-ratio of water and ore should be about 40 : 60, that is, less weight of water than of solid; and that, when crushing to produce all slime, the ratio should be the other way round, there should be a little more water than ore. Coarse sand associated with only 30 per cent of water will run down a steep launder and plainly be wet, while slime with such a percentage will be stiff and comparatively dry; such is the effect of the greater internal surface possessed by a mass completely pulverized.

Applying the above-detailed principles, cylinder mills, though all crushing by balls, pebbles, etc., in a revolving cylinder, differ sufficiently to disclose the following types: Tube-mills, Rod-mills, Ball-tubemills, and

Conical mills, through all of which the material passes as through a tube, in at one end and out at the other; and screen-faced Ball-mills, into which the material enters at one end to pass out not by the other end, but through the walls.

Tube-Mills

The prototype of this appliance is the Barrel Pulverizer, still employed in Cornwall to crush middling products arising in the concentration of tin ore, and formerly employed in California for grinding charges of gold ore. This pulverizer consists of a revolving cast-iron barrel or tube about 5 ft. long and 2 ft. in diameter, in which scrap-iron, iron balls, or pebbles are the crushing bodies, and into which ore and water are fed centrally at one end and discharged centrally at the other. In some designs the barrel is borne upon hollow trunnions through which feed and discharge take place; in others, upon rollers at the periphery (Fig. 71).

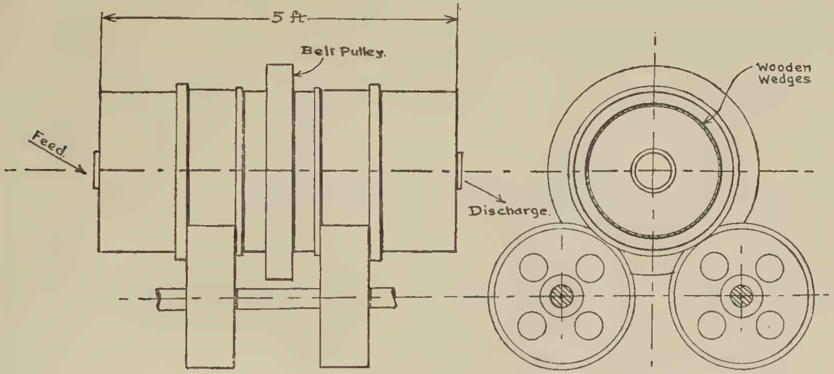


Fig. 71.

Barrel Pulverizer.—The prototype of the Tube-mill. Used since 1880 to grind intermediate products arising in process of dressing tin ores in Cornwall. The illustration shows the mill supported and driven by rollers; the original mill was supported upon hollow trunnions (p. 111).

The tube-mill of the present day is, however, an adaptation from cement grinding rather than a direct evolution from the barrel pulverizer. It consists of a riveted steel-plate tube 10—20 ft. in length and 3 ft. 6 in. —6 ft. in diameter, with flanges at either end by which it is bolted to heavy end-plates (Fig. 72). Centrally disposed in these end-plates are the hollow trunnions, one at each end, on which the mill is supported, these trunnions forming at the same time the feed and discharge openings, respectively. At the feed end usually, but latterly often at the discharge end, a large spur-wheel enveloping the end-plate constitutes the means by which the mill is driven. This spur-wheel is engaged by a pinion keyed upon the same shaft as a main driving-pulley, which pulley receives its motion by rope, belt, or silent chain from a motor or line-shaft. The silent chain avoids any slipping, but, if used to transmit the power direct from a motor, a friction clutch should be interposed, or otherwise the tremendous starting torque necessary for such a heavy mass as a charged mill might hurt the motor. Naturally, a friction clutch is also advisable if the mill is directly geared to a motor (Fig. 73).

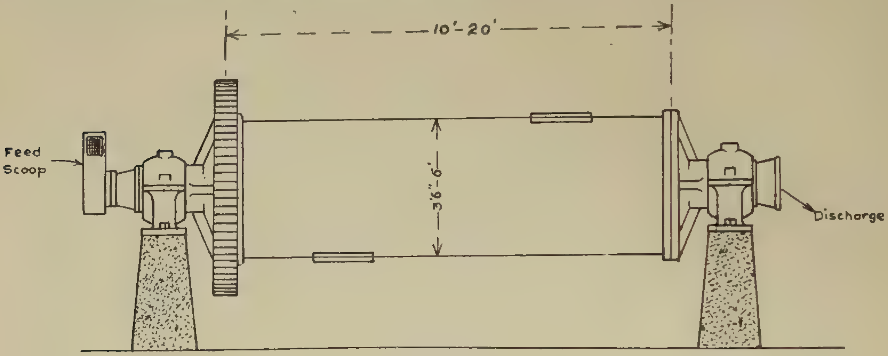


Fig. 72.

Tube-mill.—Elevation. The standard trunnion-mill is illustrated. It shows the feed scoop, hollow trunnions, end-plates, shell, manholes, and spurwheel (p. 111).

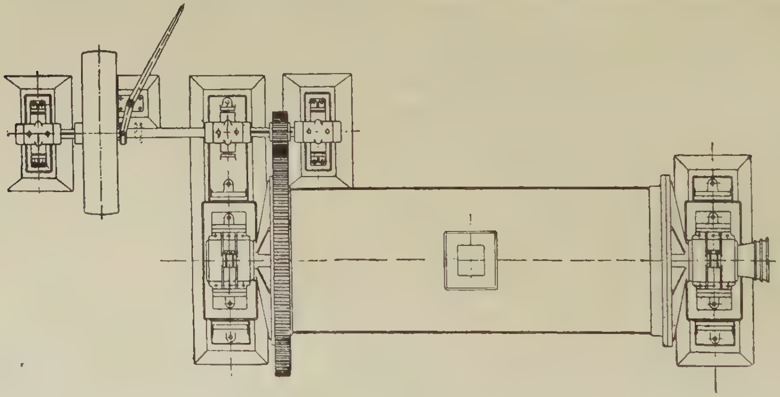


Fig. 73.

Tube-mill.—Plan. The driving pulley, shaft, and pinion are to the left hand of the mill when looking in the direction of flow; this constitutes a left-hand mill. The feed trunnion is open to be fed by the spigot of a classifier; the discharge has three projecting ribs to keep the pulp from draining back. The handle of a friction clutch engaging the driving pulley is seen (p. 112).

The method of support and driving above described is the general practice. Exceptionally, some mills are supported upon tires and driven by a toothed wheel around the body; or a mill may be trunnioned at one end and on rollers at the other. Roller support is direct and leaves the end free, but true running at the speeds and for the continuous performance demanded, is not thereby so surely obtained. Accordingly, rollers are only used for particularly heavy charges.

The material to be ground, having already been broken and crushed, is fed with sufficient water to form a thick pulp. This feed enters the trunnion either direct from the spigot discharge of a classifier, or as the delivery from a spiral scoop which, revolving with the trunnion, picks it up from a feed-box (Figs. 74, 77). This latter is now more usual, since it provides at the same time a means of feeding new crushing bodies, and permits the mill to be run more than half full should such be desired. To pass these crushing bodies onward through the trunnion this latter is sometimes lined with a spiral bushing, or it may be made with diameter increasing in the direction of the required movement (Fig. 80). When the feed is spouted from the spigot of a classifier, new crushing bodies have to be

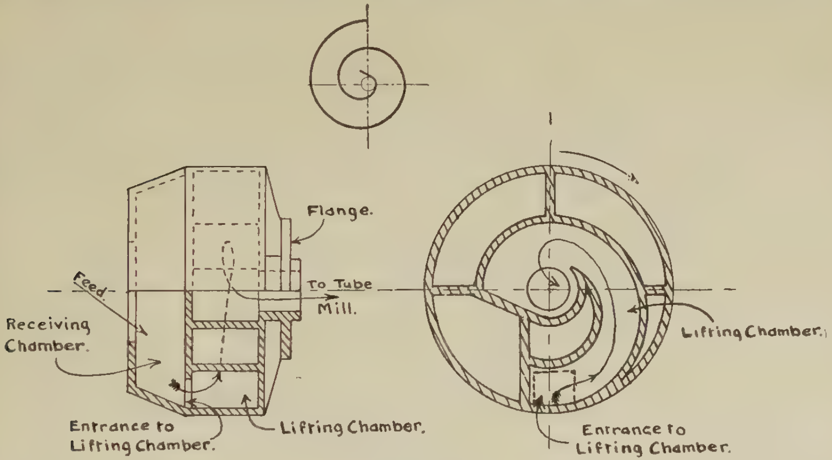


Fig. 74.

Tube-mill Feeder.—Open-fronted Scoop Feeder. This feeder is attached by its flange to the trunnion of the tube-mill. The feed, to which crushing bodies can be added, is fed into the receiving chamber. During revolution the entrance to the adjoining spiral lifting-chamber dips under the accumulated feed permitting some to pass within; whereafter it is raised by the spiral scoop to the central opening through which it passes into the mill. For feeding crushing bodies and coarse ore this feeder is better than the ordinary scoop, since there is no feed-box against the side of which such material might become jammed; moreover it needs no lip to dig into water-packed material and in so doing to wear out (p. 113).

fed through the trunnion, or the mill must be stopped to introduce these bodies through a manhole. To pass them through the trunnion necessitates an open trunnion, that is, one free from grate or stuffing-box; the mill cannot then be charged above the pulp level. An additional advantage of the scoop feed is that the material is lifted from below the axis of the mill, and height is thereby gained.

Discharge at the other end is central and usually by simple overflow, the rate being determined by the rate of feed. To facilitate this discharge the opening at that end is generally of somewhat larger diameter than the feed opening, a gradient being thereby created, common diameters being 8 in. and 10 in. respectively. When it is desired to work with a charge above the axis of the mill, the discharge opening is closed with a grate having holes small enough to retain crushing bodies of useful size, but large enough to permit the discarding of those too small to be of use (Fig. 75). Even with a low charge a screen plate is often used for this latter purpose; or within the trunnion itself a reversed-spiral lining is inserted.

In crushing cement the tube-mill works dry, when, ordinary overflow discharge being impossible, discharge from the crushing space takes place through a grate or diaphragm occupying the entire cross-section of the mill. The material passing this diaphragm finds itself in the narrow space between it and the end plate, in which space are radially-disposed scoops which, as the mill revolves, lift the material to the hollow trunnion. Such a discharge is described as peripheral, though really it is scoop discharge. Scoop discharge is not peculiar to dry crushing nor to the crushing of cement clinker, but is largely used in the wet crushing of ore, both with tube-mills and with ball-mills (Fig. 78). It possesses the advantages of effecting a more positive discharge and of lowering the water pool; on the other hand, it introduces parts on which a good deal of wear comes, and also results in so lowering the pulp level at the discharge end that some of the pebbles may be out of action. Exceptionally, with dry material, true peripheral discharge through apertures in the periphery at the discharge end is seen.

The crushing bodies employed are generally flint pebbles, from which fact the appliance sometimes takes the name of "pebble-mill." These pebbles are well rounded pieces of flint or other hard rock, 2½—4 in. in diameter. Suitable pebbles are found on the shores of many lands, those coming from Denmark, Newfoundland, Greenland, and Northern France, being particularly in request. For mines well situated in respect to transport the cost of such pebbles delivered is not great, but far inland it might well be prohibitive. In these remote situations suitable pieces of hard ore may serve as pebbles. The consumption of such pieces is naturally many times greater than that of ordinary pebbles, and for lack of initial roundness they are not so efficient; but being of ore they gradually add their own weight to the amount crushed, and no hurtful drop in mill capacity occurs.

Small steel balls, or balls of an iron-manganese composition, are also employed, though the fact that ordinarily tube-mills are applied to prepare material for cyanidation—in which the presence of finely divided iron might be detrimental—has perhaps militated against their use. Such balls offer the advantage that, being much smaller than flint pebbles of the same weight, they provide more points of contact in the same volume of charge; accordingly, a smaller tube-mill using steel balls would be as effective as a larger mill using flint pebbles.

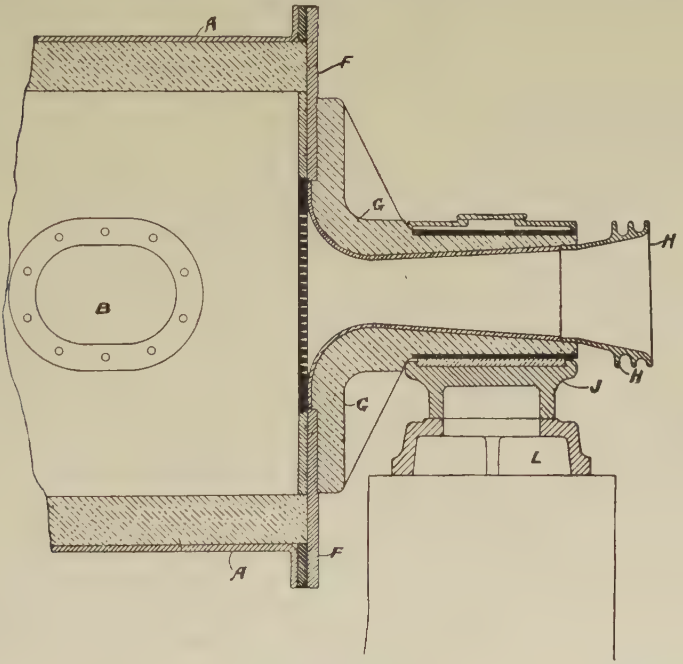


FIG. 75.

Tube-mill.—Section at Discharge End. This section shows the grate fixed across the central discharge to keep the crushing bodies from ejection. This grate is placed in position from the inside of the mill, entry being obtained through the manhole B. The hollow trunnion with diameter expanding in the direction of discharge is clearly shown. Beyond the actual trunnion the discharge is continued by an additional piece provided with three ribs, H, to keep the pulp from draining back into the bearing. The illustration further shows a flint or silex lining (pp. 114, 117).

Small hollow steel or iron cylinders have also been tried, but not adopted. The idea of a cylindrical crushing face is, however, attractive, because lines of contact then take the place of points and greater opportunity for crushing is presented. Following this idea, heavy steel rods extending the whole length of the mill have been found very satisfactory where a particularly fine product was not desired. Rods, unlike balls, having one dimension continuous from end to end of the mill, permit a heavier charge to be accommodated in any given mill; in addition, rolling on one another there is little chance for any particle to escape uncrushed. Rod mills accordingly have great capacity for their size. The rods, however, are liable to become bent when worn, while compared with balls they are always awkward in handling. Round crushing bodies, accordingly, are still preferred; they possess the advantage that turning continually in all directions, the whole surface is available for crushing and remains true, no grooves or depressions developing.

To maintain the number of points of contact as high as possible, mixed sizes of crushing bodies should be present. Such mixed sizes need not, however, be fed specially, since they arise normally by wear from bodies of maximum size fed at regular intervals. On the other hand, pebbles of greatly diminished size are useless and should be discarded. The consumption of flint pebbles per ton of original feed varies with the hardness of the ore, the size of the feed, and the fineness to which comminution is taken. It is usually between 2 lb. and 5 lb. per ton ground.

Concerning the charge of crushing bodies, without a grate this cannot half fill the mill. But with a grate at the discharge end and a scoop at the feed end it may rise above the mill axis, giving better balance to the machine and diminishing the demand upon power. Crushing fine material a well-filled mill more than maintains its capacity even though the opportunities for impact are obviously diminished; the limit, however, appears to be passed at two-thirds the internal volume.

A charge weighs anything from 4 to 15 tons, depending upon the mill size. Speaking generally, for mills of similar homologous dimensions, capacity is proportional to the weight of charge.

To protect the steel shell of these tube-mills against wear, an effective lining is necessary. In cement grinding this lining consisted of iron or steel plates an inch or so in thickness, bolted through the shell; covered by a dry skin of crushed material this lining was effective and lasted for years. But scoured by wet pulp in the grinding of ore the life of such a lining is not usually more than six or eight months; moreover, when harder metal is used the surface becomes so polished by wear that the charge slips and crushing suffers. A "smooth or plane lining" is only preferred for very fine grinding, where the greater force of impact is not required.

Flint or silex blocks of brick size, set on edge in cement, have proved satisfactory in general use, the surface being relatively rough. An objection is that the relining of such mills puts them out of commission during the forty-eight hours or so the cement takes to set, whereas a new steel lining can be inserted in a few hours. A second objection lies in the reduction of the mill diameter by so thick a lining, and the consequent difference in capacity and power-demand between the newly-lined and worn conditions of a mill. The flint lining is, nevertheless, simple and effective; it has a life of six to eight months (Fig. 75).

To moderate the first objection, the time taken in relining may be made reasonable by setting the blocks previously in cast-iron grids of convenient size and suitably curved to fit the interior. Such filled grids permit a more rapid assembly into place. With intervening webs across them, and using mine rock instead of the more expensive flint blocks, a suitable rough lining of honeycomb pattern is cheaply produced; such a lining has a similar life of six to eight months.

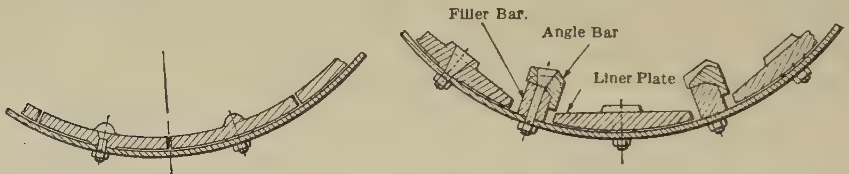
Giving a firmer grip upon the charge a number of "ribbed linings" are now in use. The earliest of these was a lining cast with longitudinal grooves in which the pebbles eventually became firmly fixed, making a lining of themselves. This lining, the El Oro, is effective, and has a life of one to two years, depending upon the metal used, manganese steel giving a long life (Fig. 76).

A more positive lift to the charge is given by longitudinal bars or lifters spaced regularly around the interior, with flint bricks or metal plates between. Or, metal plates cast with a rib on them may be so set in the mill that the ribs are continuous from end to end (Fig. 76).

Ribbed linings are of best service where impact is desired, that is, with coarse material; for the same class of work they allow a somewhat slower speed. Plane linings, on the other hand, are good with fine material and for the production of slime.

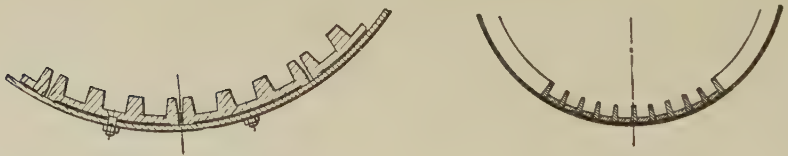
Tube-mills are employed to make fine sand or to reduce all to impalpable slime. In South Africa, notably, the first of these policies is pursued; on such other gold and silver mines as have their valuable content intimately distributed throughout the gangue, the second is the policy. Herein lies the reason of great differences in tube-mill capacity. A full-size mill may be taken to be one of 20 ft. length and 5 ft. diameter. Such a mill is capable of crushing 250 tons of original feed per day from 3 mesh to the condition of fine sand, whereas when reducing 16-mesh material to slime its capacity would be only about 100 tons per day. Smaller mills having similar relation of diameter to length, have capacities in proportion to their smaller internal volume. It is the diameter which determines the maximum size of material the mill can conveniently handle, and the length which determines the quality, in point of fineness, of the product. While maintaining the diameter of 5 ft., shorter mills, 16 ft. in length, are now considered standard in South Africa.

Though their greatest application is perhaps with precious-metal ores, tube-mills of short length have to some extent also been used in regrinding base-metal ores for further water-concentration; at Broken Hill, New South Wales, for instance, they have replaced pans for that work. When the length, still further shortened, becomes of similar dimension to the diameter, the opportunity is generally taken to increase the diameter and to use relatively-heavy steel balls. Such mills, belonging to the ball-tubemills next to be described, are known as "granulators," because of the granular nature of their product.



Ribbed Plate Lining

Komata Lining



El Oro Lining

Bar Lining.

Fig. 76.

Tube-mill Linings.—The ribbed-plate lining is simple; the ribs extend from end to end of the mill. The Komata Lining has lifting bars—specially protected by manganese-steel angle-bars—alternating with independent plates. These bars project four inches or so into the mill interior, so that they exert a pronounced lift upon the charge. Into the flutes or grooves of the El Oro lining, pebbles fix themselves to form a surface over which the charge does not readily slip. A similar lining is the Bar Lining, the projecting bars of which are kept in place by others laid flat against the mill interior, the spaces between the bars being filled with broken rock cemented into place (p. 117).

The power required by the tube-mill is relatively high, being about 10 h.p. per ton of pebble charge, a mill of full size requiring 100 h.p. continuously and 150 h.p. at the start. Taking the former figure and the figures of capacity given above, the capacity when grinding to fine sand

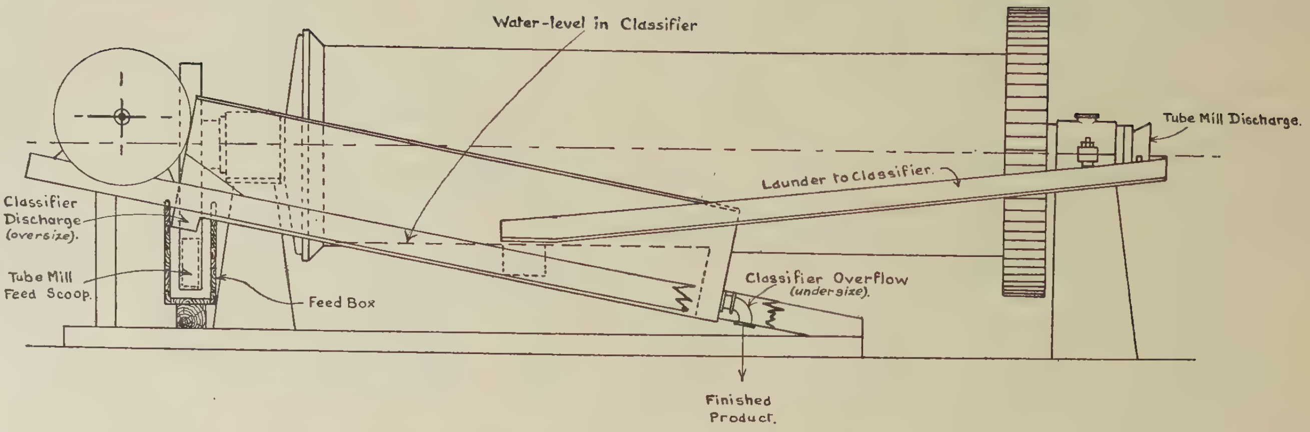

Water-level in Classifier
Tube Mill Discharge.
Launder to Classifier.
Classifier Discharge (oversize).
Tube Mill Feed Scoop.
Feed Box
Classifier Overflow (undersize).
Finished Product.

FIG. 77.

Tube-mill.—In Closed Circuit with Mechanical Classifier. A long tube-mill is shown driven from the discharge end, which leaves freedom at the other end for feed-box, classifier, etc. Into the feed-box everything is brought, original feed, return feed, and crushing bodies. All these the ample scoop picks up, delivering them through the feed trunnion. At discharge the pulp passes along a launder into the drag-classifier, from whence the finished product overflows, while the sand is dragged up out of the water at the upper end of the classifier and delivered into the feed-box (pp. 109, 113, 120, 267).

about 200 lb. per h.p. hour, and when grinding to slime about 80 lb., the ratios of size-reduction being about 15 and 10 respectively. These figures illustrate the greater consumption of power in fine reduction.

Considering the large capacity of these tube-mills the labour and attention required are not great; the mill has all the advantages of a large unit. In addition, the absence of screens, screen frames, scrapers, and small parts generally, reflects favourably upon the operating cost. Though, therefore, power consumption is high, the total cost of crushing by this mill, seeing that it operates at the fine end of the crushing system, is relatively low, and may be taken to be about 9d.—12d. per ton when grinding to fine sand, and 2s.—3s. per ton when grinding to slime. No other machine can so cheaply do this class of work, nor does any other machine compare with it in the reduction to slime. In regard to first cost, this mill has the disadvantage of high prime cost for itself, for its foundation, and for its means of driving, but over a long life this heavy initial expense is more than made good by the relatively low operating cost.

For efficient running tube-mills require proper speed, adequate pebble-charge, a good rate of feed, and right dilution; invariably, also, they work best in "closed circuit" with a classifier (Fig. 77). When the speed is too high or the feed too low the mill will roar, whereas when the conditions are proper it rumbles, emitting a lower and more continuous note.

Ball-Tubemills

Crushing by balls in a revolving tube has also been applied to coarse material, such, for instance, as the breaker product. To accomplish such work the balls have to be heavy, and are of steel, forged rather than cast; the mill diameter must be maintained or even increased to give the balls the greater fall necessary; the mill must be short, because, if long, the balls at the discharge end would find little useful employment; while, finally, a grate diaphragm and scoop discharge are necessary generally, since the product may be too coarse for the ordinary overflow (Fig. 78).

Such mills as these, under the name of Grondal and Ferraris mills, were used in metalliferous mining before the introduction of the tube-mill, but received no extended application. To-day, however, with better knowledge resulting from the successful employment of tube-mills, and with better appliances for operating in closed circuit, a greater application of these ball-mills is probable (Fig. 81).

Recent mills are 5—7 ft. in length and of equal or even greater diameter, supported as usual on feed and discharge trunnions, and run at a speed sufficient with the help of ribbed lining to develop the necessary

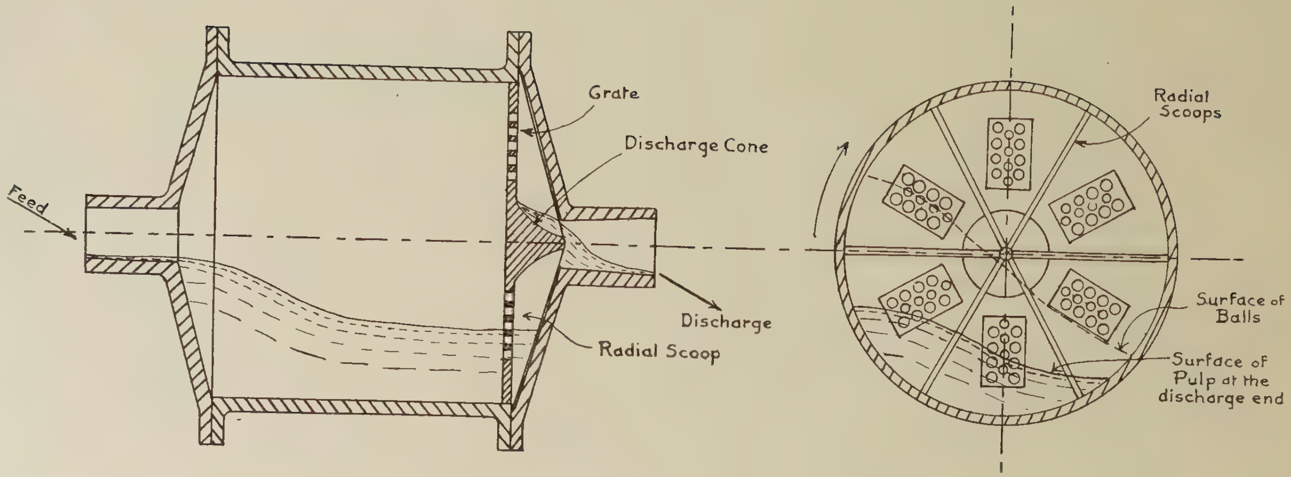


FIG. 78.

Ball-Tubemill.—Diagrammatic Section. This diagram illustrates the working of peripheral or scoop discharge. The scoops are shown radial, though they are often curved vanes leading to the centre (pp. 110, 114, 120).

impact. The balls when new are 5—7 in. in diameter and weigh 20—50 lb.; when worn to about $1\frac{1}{2}$ in. they are discarded. A charge of such balls weighs from 5 to 12 tons and nearly half fills the mill. The lining is of steel plates ribbed to give greater lift to the balls, the ribs being often so arranged that the charge in rolling back descends a series of steps, hence the term "stepped lining" (Fig. 79). The feeder is generally of the 'combination' type, arranged to take dry material and balls at its open centre, and to dig-up returned sand from a feed-box (Figs. 80, 81). Usually a grate or diaphragm near the end separates a small compartment in which radial scoops lift the product to the central discharge as the mill revolves. By some this grate is considered to block the free discharge so necessary for quick passage through the mill, and the overflow discharge is preferred. In an exceptional design this end compartment does not exist, but the discharge is directly through a screen plate covering that end.

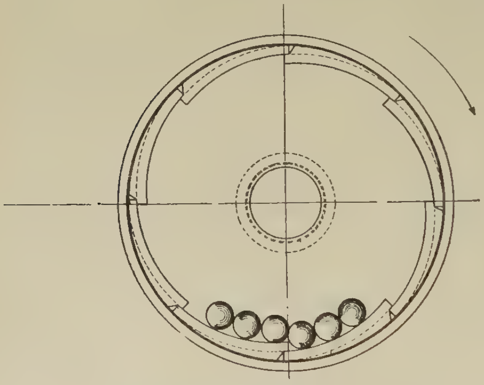


FIG. 79.

Ball-Tubemill Lining.—Stepped Lining (p. 122).

A particular mill of this type, the Marcy mill, came into prominence through its adoption at one of the large disseminated copper mines, the Inspiration, Arizona, where, working in closed circuit with a mechanical classifier and without further help, this mill crushes the breaker product to about 40 mesh, suitable for flotation concentration. The capacity of an 8-ft. mill of this type doing such work is about 450 tons per day, while the product, owing to the quickness of the discharge, is uniform and fairly granular (Fig. 81).

When these ball-tubemills are employed to regrind middlings or tailings in preparation for further water-concentration, or to do similar work where reduction in size without undue production of slime is desired, they are often described as "granulators." Such mills come midway between tube-mills proper and ball-tubemills (Fig. 82).

In general the power consumed by these mills is high and somewhat greater per ton of charge than with ordinary tube-mills, but their capacity being more than proportionately large they are efficient. A 7-ft. mill would require about 140 h.p. and crush about 280 tons per day from 2 in. to 20 mesh, figures equivalent to about 170 lb. per h.p. hour, and a size-reduction of 80; the capacities for smaller mills would be in proportion to their smaller internal volumes. At such a ratio of size-reduction these mills are at times sufficient in themselves to complete the necessary comminution of an ore. At other times they accomplish the first stage of a reduction which is completed by tube-mills.

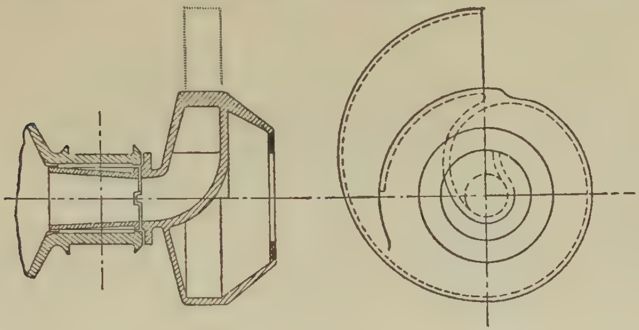


Fig. 80.

Ball-Tubemill Feeder.—Combination Type. With this feeder coarse original material passes in through the open front, while returned material is scooped up from a feed-box into which it has been delivered by the classifier (pp. 113, 122). Hines, *Trans. A.I.M.E.*, vol. lix., 1918, p. 259.

With a large capacity and operating at the coarse end of the crushing system, the consumption of balls is rather heavy, being about 1·5—2 lb. per ton, the wear of the steel lining being about one quarter that figure. On the other hand, the operating cost is low, being about 6d. per ton, and the machines occupy little floor space.

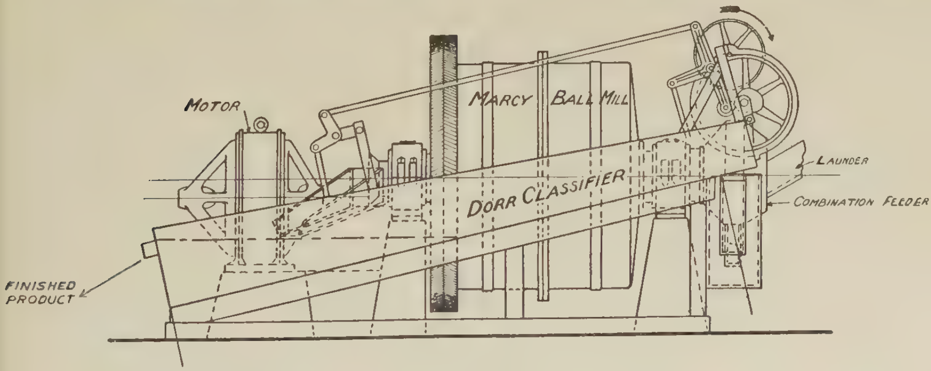


Fig. 81.

Ball-Tubemill.—Working in Closed Circuit with a Mechanical Classifier. The mill illustrated is driven at the discharge end to make room for the classifier and feeder at the feed end (pp. 109, 120, 122, 267).

To do good work they must be fully and regularly fed with original material by an independently-driven feeder; the size of the balls must be suited to the size and hardness of the ore; and the ore may not be both large and hard.

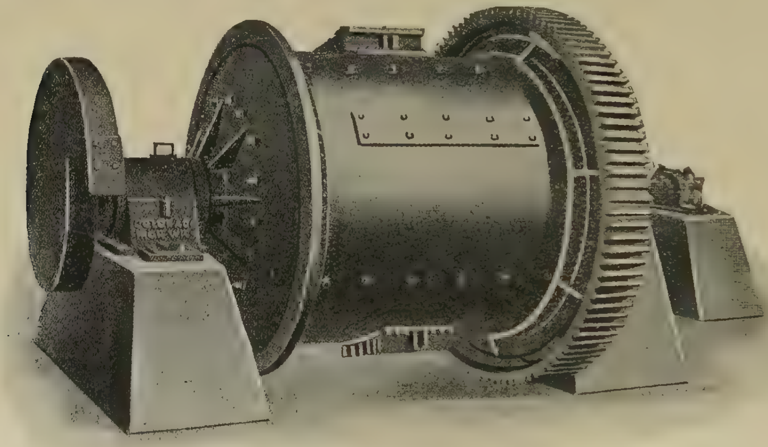


Fig. 82.

Ball-Tubemill or Granulator.—General Appearance. This illustration shows particularly well the ample scoop-feeder sometimes necessary (p. 122).

Conical Mills

The tube-mills which have so far been described have been cylindrical and the crushing conditions have been fairly uniform from end to end. Seeing that the material being crushed enters in a relatively coarse condition to become progressively finer, it becomes a question whether such uniform crushing conditions could be altered the better to suit the progressively finer work. With this idea a conical mill, known as the Hardinge mill, was evolved, in which two hollow cones, one flat and the other relatively steep, are joined at their bases to the ends of a short cylinder; at the apex of the flat cone is the feed entry, and at the other apex the discharge, the mill being supported upon hollow trunnions exactly as the ordinary tube-mill (Fig. 83). The crushing bodies are pebbles or steel balls as before. In operation some of these are large and new, while others are worn and small, the bulk being of an intermediate size. In a cylindrical mill all these would arrange themselves, uniformly from end to end, with the larger ones at the periphery and the smaller ones towards the centre. In a conical mill, on the other hand, the large pebbles or balls, in their endeavour to hug the periphery, fly to the largest diameter, that is, to the cylindrical portion, from which point to the discharge the crushing bodies will be of continually decreasing size. The feed cone being flat, the material quickly arrives in the cylindrical portion, where it receives attention from the maximum forces, whence it passes towards the discharge, becoming on its way subject to smaller crushing forces conceivably more suitable to its own diminished size, till finally it overflows as a finished product. The continued entry of the feed determines the rate of this passage, which, as a rule, is further promoted by inclining the axis of the mill slightly downwards in the direction of discharge. It is claimed that this mill makes better use of the charge than ordinary tube-mills, in which the large pebbles near the discharge are not doing the work for which by their size they are fitted; and that in consequence a smaller charge suffices and less power is consumed.

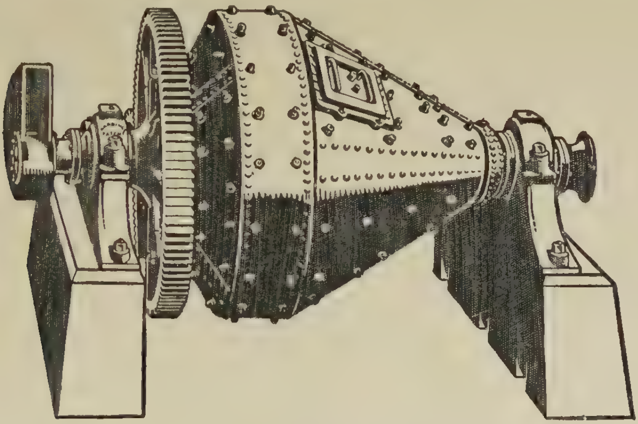


FIG. 83.

Conical Mill.—The Hardinge Conical Mill (p. 124).

Whatever the value of this advantage, in practice the maximum diameter of a conical mill is larger than the diameter of a cylindrical mill of equal capacity, and the extra power required to give the greater peripheral speed which greater diameter brings, must be set against that advantage. A 5-ft. cylindrical mill 16 ft. long, running at a peripheral speed of about 450 ft. per minute, would probably have to be replaced by a conical mill of 8 ft. maximum diameter, which at a proper speed of revolution would give a peripheral speed of 570 ft. per minute; from that maximum the speed would fall to a minimum at discharge.

A proper test between the short tube-mill and the conical mill has yet to be recorded. These two types both appear to be efficient machines and to be capable of doing similar work. They are similarly lined, use similar crushing bodies, and are subject to similar wear. The ordinary mill has the advantages which come from a more simple design, while the conical mill facilitates discharge by gradually leading up to it. The latter has, moreover, proved itself to be an efficient granulator giving a uniform product, the exact nature of which varies with the length of its cylindrical portion; the longer this length, the finer the product. As such it has received the endorsement of wide adoption. The conical mill is, however, never a slime-producing machine like the tube-mill.

As with cylindrical mills, steel balls are used in conical mills principally for crushing coarse material and flint pebbles for fine material. It is, however, with the latter that the conical mill has been notably successful. With coarse material it appears probable that inadequate work is accomplished in the conical portion, where diminished size and diminished velocity likely render the crushing bodies impotent to continue the work.

(SCREEN-FACED) BALL-MILLS

These mills, though fed at one end like the ball-tubemill, do not discharge at the other, but through apertures in the walls. They are enveloped by screens, and are usually applied to the dry crushing of coarse material. The Krupp ball-mill is representative of this type (Fig. 84).

In detail the screen-faced ball-mill consists of a drum-shaped cylinder, 7—9 ft. in diameter and 3·5 -4·5 ft. in width, fixed on an axial shaft laid horizontally, and revolved at a speed of 20—25 r.p.m. The sides of this drum are of steel plate bolted to naves upon the shaft and to a flange around the drum, this drum being of cast steel. The interior surface of this casting is not perfectly cylindrical, but made up of a number of steps known as grinding plates, down which the balls cascade as the mill revolves. These plates, six to twelve in number, slightly overlap one another, each in succession slipping under the next in front. Just before being thus overlapped, in each grinding plate are discharge holes of coarse size, while the rectangular passage between it and the plate overlapping is shut by a strip perforated with holes of similar size. Outside the drum comes a circle of punched screens, and then a concentric envelope of fine screens attached to screen frames. The circle of punched screens is not continuous but broken at return scoops which, extending from the fine screen to each grinding plate, permit the oversize from both screens to pass back into the drum, this taking place as the mill revolves. The entire mill is finally surrounded by a sheet-iron housing terminating below in a pyramidal pocket leading to discharge, and connecting above with a fan by which the fine dust is drawn away.

The crushing bodies are steel balls, which, when new, are usually 5 in.

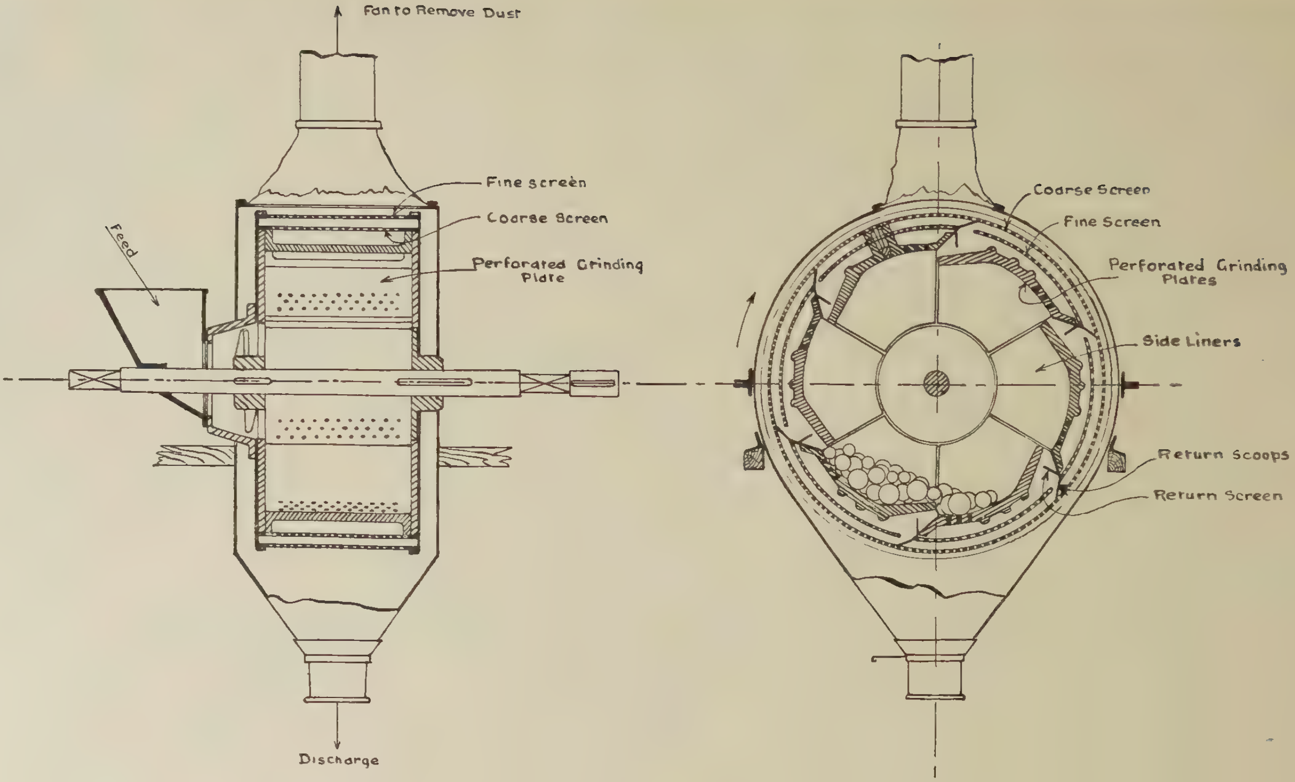
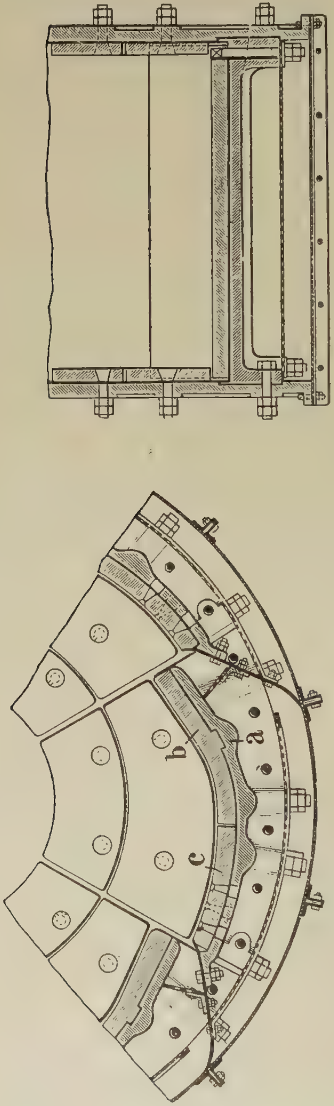


FIG. 84.

Ball-mill.—The Krupp Ball-mill (p. 126).

in diameter and weigh 18 lb., though balls of 7 in. diameter and 30 lb. weight are sometimes used, the size of ball depending upon the size and character of the material undergoing comminution. A charge of these balls generally weighs 1—2 tons, sufficient when the mill is run at a normal speed to climb nearly to the level of the shaft, before falling back. The balls therefore are not overthrown, but the stepping of the grinding plates gives them opportunity to drop and thereby to develop the necessary impact; in the larger mills this drop is about 5 in. In addition, there is a good deal of rolling and rubbing between the balls, during which the comminution begun by impact is completed. To maintain efficiency the balls must not be allowed to wear smaller than about $1\frac{1}{2}$ in., and from time to time the charge is taken out in order that the uselessly small balls may be discarded. Ball wear takes place at the rate of about 0·5 lb. per ton crushed.



a Mantle Plate b Hunch Plate c Perforated Plate

Fig. 85.

Ball-mill Details.—Details of Krupp Ball-mill (p. 128).

Almost equal wear comes on the interior of the mill, that is, upon the sides and the grinding plates. To provide for this the grinding plates are made in three pieces, one being the base or mantle plate, and the other two respectively the hunch plate and the perforated plate, these being the renewable wearing surfaces (Fig. 85). The sides likewise are protected by liners shaped to the circular surface they cover, and bolted into place. As a rule, the wearing plates last about 6 months and the side liners a year.

The size of the material to be crushed varies from 2 to 5 in. and is generally about 3 in. This material must be quite dry, that is, containing not more than about 1 per cent of moisture, or otherwise the screens will tend to clog and the capacity will be seriously diminished. It is fed through a hopper on one side, the nave on that side having a central opening in which spiral blades are fixed to ensure entry and to turn back any material which might otherwise be thrown out. For convenience in feeding, the mills are best arranged in pairs consisting of a right-hand mill and a left-hand mill, so that the two naves come together. They are fed by uniformly-running independently-driven feeders, it being difficult to arrange a feeder which shall operate only when more ore is definitely

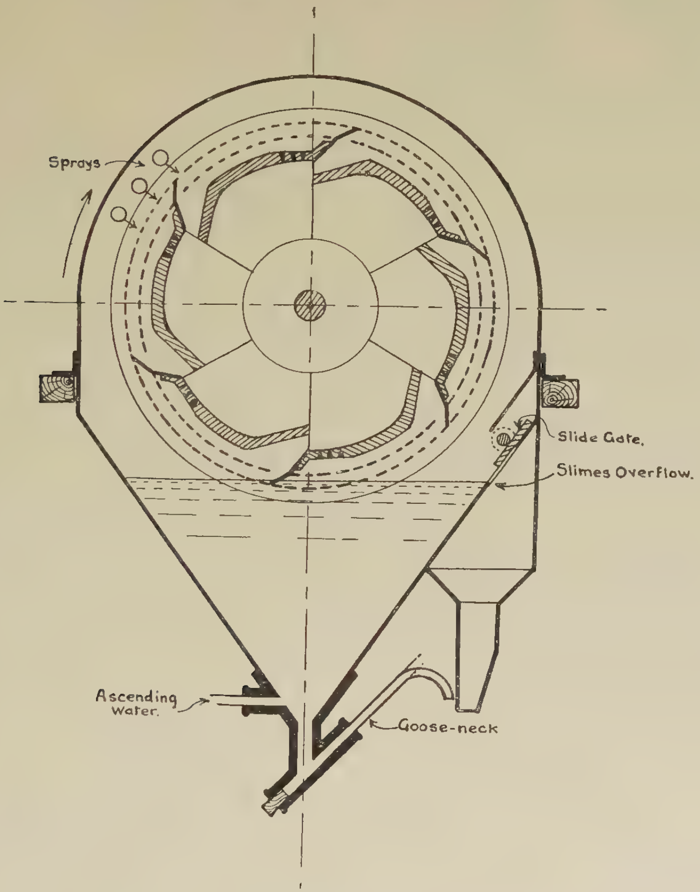


Fig. 86.

Wet Ball-mill.—Krupp Mill arranged for wet crushing (p. 130).

required. The attendant has to judge by sound whether the mill is being fed properly; an empty mill roars, a mill properly fed rumbles and when opened no balls are to be seen, all being embedded in ore.

With respect to the work they accomplish and their capacity, crushing 3-in. material these mills generally deliver their product through a 20—30-mesh screen, making a size-reduction of 120—180. Owing principally to the generous screening area, which is usually from 60 to 100 sq. ft., the capacity is high, generally 40—100 tons per day, and the product is granular. Consuming 20—50 h.p. the amount crushed per h.p. hour is about 180 lb.

These mills were largely used at Kalgoorlie, where dry-crushing was necessary owing to subsequent roasting; there the cost was about 2s. per ton. In addition, the mill has been largely used on the Continent, where, though the general design remains the same, different makers have introduced slight modifications, one consisting in discharging on both sides near but not actually at the periphery, others having for object the adaptation of these mills to wet-crushing.

Though this ball-mill is particularly employed in dry-crushing, it has also been used wet. Crushing even then is not done in water, but jets of water play inside while others play upon the screen from outside. The water and crushed material fall below into a conical pocket from which there is a bottom discharge of the sand and an overflow of the slime. In this pocket the level of the water is such that the fine screens dip below it as they revolve, becoming cleaned thereby (Fig. 86).

Such wet-crushing can, however, only be used with moderately soft ore, or the wear of balls, plates, and screens would be excessive. The capacity is favourably influenced by the presence of water.

STAMPS

Under cylinder mills the advent of impact as an important force-application in crushing was recorded; in those mills impact-crushing was accompanied by grinding. With stamps, grinding is finally abandoned, leaving impact-crushing supreme. The ore lies upon a die in a mortar, and impact arises either from the free fall of a Gravity stamp; or from the modified fall of a Crank-lifted stamp; or from the forced fall of a Steam stamp.

Gravity Stamp

With gravity stamps a weight W is lifted through a height h and allowed to fall through that height. The energy available for crushing is the kinetic energy of the falling mass, that is, $Wv^2/2g$; or since $v^2=2gh$, $W2gh/2g$; or Wh. That is to say, the energy available for crushing is equal to the work done in lifting the weight. The force available depends upon the distance moved through while this energy is being expended; if that distance be small, the force will be correspondingly great. With a single piece of ore upon the die this distance is the amount of deformation suffered before the elastic limit of the piece is passed. This deformation is obviously very small indeed, even when given time to develop fully; but under the smartness of the blow it is still smaller, making the force available for crushing still greater. Ordinarily, however, there will be upon the die not one piece only but many, and the energy of the stamp will be partly absorbed in rearranging these pieces; the stamp accordingly will sink deeper before being brought to rest and its crushing force will be proportionately diminished. When this condition is pronounced the blow is said to be "cushioned."

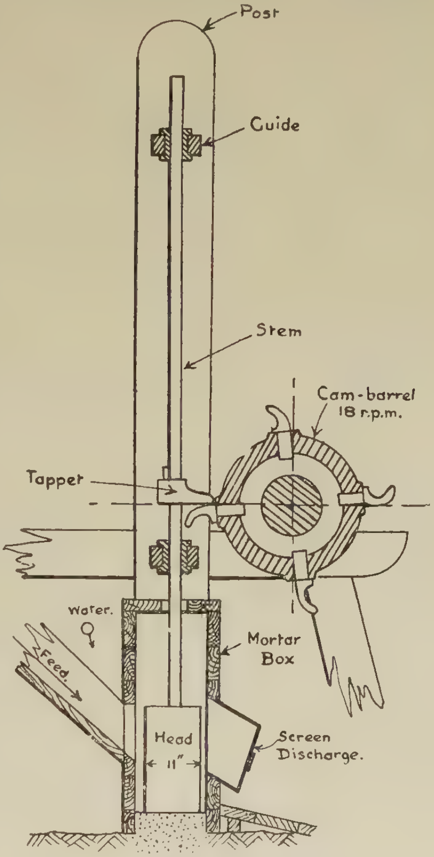


Fig. 87.

Cornish Stamp.—This stamp is rectangular, the stamp heads being about 11″ × 6″ in section and 20″ in length. Four stamps are usually in each box, and any number of boxes constitutes a battery. These stamps are usually driven from a beam engine working in the centre of the line of stamps, though in stream works, from which the above illustration was taken, they are generally driven by water wheels (p. 131).

Cornish Stamp.—The original type of gravity stamp is represented by the Cornish stamp (Fig. 87). In this stamp the pestle or falling weight consists of a rectangular stem of wrought-iron about 12 ft. long and $2\frac{1}{2} \times 4$ in. in section, and a rectangular head of white or mottled iron about 7 × 11 in. in section and about 20 in. long, this head being cast around the lower end of the stem. The mortar is a wooden box large enough to contain four of such heads. The bottom of this box is formed either by a long cast-iron die extending from end to end, or it consists of ore which has been beaten hard into position. At the back is a feed opening through which a chute delivers the ore to be crushed, while, in front, screens are fixed to determine the size of the discharge, these screens being usually of punched copper plate. Water to the extent of about 6 of water to 1 of ore is fed with the ore.

Each mortar box stands between two upright posts across which two guide bars extend horizontally, one not far above the box and the other near the top of the posts. Through proper ways in these guide bars the stems pass, being thereby maintained vertical.

The stamps are lifted by cams upon a barrel revolving in front, these cams engaging tappets upon the stems. Of these cams there are four or five in the vertical circle, each engaging the same tappet in turn. The cam barrel makes 12—15 r.p.m., and accordingly about 60 blows or drops are made per stamp per minute.

In this conversion of circular motion to vertical lift there are one or two conditions to satisfy. Firstly, the lifting force must be vertically directed and horizontally received, so that no needless side-thrust comes upon the guides; and secondly, the lift shall be regular and known.

These two conditions are satisfied by making the cam face an involute to the circle having as radius the horizontal distance between the barrel axis and the contact between cam and tappet (Fig. 88). An involute is such a curve as would be described by the end of a piece of stretched string in process of being normally unwound from a stationary drum, the drum circumference being then the pitch circle of the involute; the relation between involute and pitch circle is such, that a tangent to the circle if drawn to cut the involute would be at right angles to the involute tangent drawn at the point of cutting. But the line of lift is a vertical tangent to the pitch circle chosen, and the lifting force acts vertically upward along that line; accordingly, the horizontal face of a tappet capable of displacement along that line of lift would be tangential to an involute cam at all points of the lift, and no side-thrust would develop.

A second consequence of the natural relation of involute and pitch circle, is that the tangent at any point of the circle drawn to cut the involute has always the same length as the arc around the circle from that point to the origin of the involute. While, then, a given circular movement were taking place around the circle, an equivalent but vertical movement would be forced upon a tappet engaged by an involute cam. The involute cam thus brings regular and known lift.

During lift, the point of the cam in contact with the tappet moves regularly along the cam surface in the direction from origin to extremity. Any desired lift can, accordingly, be obtained by bringing the cam to an end when the circle tangent is equal to the desired lift, provided that lift began at the origin of the involute, that is, from the plane containing the axis of the pitch circle. If lift begins at a plane above that axis, the length of cam face must then provide for the further lift to that plane. Lift cannot begin below that plane.

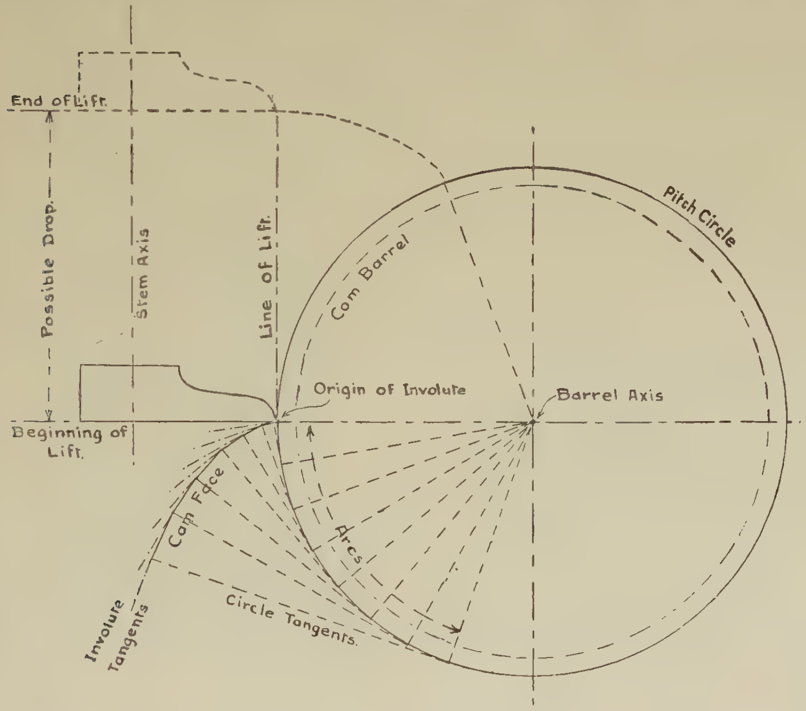


FIG. 88.

Relation of Cam Barrel, Pitch Circle, Cam Face, Tappet, and Lift, of a Cornish Stamp.—To obtain the cam face necessary for a given lift, the pitch circle is first drawn with radius stretching from the barrel axis to the toe of the tappet. Around that circle the height of lift is then laid as an arc, which is readily done, the length of the whole circumference being known. The point from which this height is measured will be the origin of the involute cam face, where lift begins. At the point to which it extends, a tangent to the pitch circle is drawn with a length equal to the height of lift; the end of this tangent will mark the end of the cam face. When in due course that tangent becomes the vertical tangent it will lie in the line of lift and be the measure of the complete lift. Intermediate points are obtained by taking half, quarter, three-quarter, and other fractional lifts. Allowance must be made for the fact that in practice lift hardly begins at the origin of the involute (p. 132).

The cam barrel is generally hollow and of cast-iron, the cams fitting into holes in which they are keyed or wedged. One barrel generally serves four boxes or sixteen stamps, and, to equalize the stress which comes upon it, the cams are so placed that the stamps fall one after the other, beginning at the first and ending at the sixteenth. The cams engage the tappets end on, and the stamp accordingly is in the same plane as the succession of cams which operate it. They are generally designed for a maximum possible lift of about 10 in., having which, the actual lift may be made less by placing the tappet higher on the stem. During lift the stamp is guided between the two guides already mentioned, one being below and the other above the tappet.

With respect to the crushing force, if the falling weight were 600 lb. and the drop 10 in. the energy available would be 500 foot-pounds, which, if expended while a distance of $\frac{1}{2}$ in. were traversed, would represent an average crushing force of 12,000 lb.; or 24,000 lb. if the distance were $\frac{1}{4}$ in. In practice it is found that the force developed is sufficient to deal with the product from the breaker. Crushing broken hard ore to 20—30 mesh and thereby effecting a size-reduction of about 150, each stamp, consuming about 1·5 h.p., will reduce about 1 ton per day, these figures being equivalent to about 60 lb. per h.p. hour. The capacity of a stamp in tons per day of twenty-four hours is described as the "stamp duty."

The wear on the cast-iron head is considerable, being about 2 lb. per ton crushed. The back of the head, where the ore enters, wears more than the front, making the crushing face oblique. Before this irregularity becomes too pronounced the stamp is turned back to front, when wear brings the face to a normal condition again.

Californian Stamp.—The Californian stamp differs from its Cornish prototype chiefly in having a circular instead of a rectangular cross-section, this design permitting its rotation around its own axis. As will be explained later, this rotation is brought about during lift by the friction between cam and tappet, the former engaging the latter not end on, as with the Cornish stamp, but to one side.

The "stem" of this stamp is a circular mild-steel shaft slightly tapered at each end. In diameter it varies from 2·5 to 4·5 in., and in length from 10 to 16 ft. It weighs generally from 300 to 600 lb., representing 35—45 per cent of the total falling weight. The "head" into which one end is inserted is a cylindrical iron or steel casting, with a conical socket at the top to receive the stem, and another at the bottom to receive the shank of a replaceable piece known as the shoe (Fig. 89). At the end of each socket is a drift-way for convenience in driving out stem or shoe, the former being held by its good fit, a driving fit, and the latter by small wooden wedges around it. The diameter of the head is 8—9$\frac{1}{4}$ in., while its length varies from 18 to 40 in. according to the desired weight of the stamp. In recent mills, increase of stamp weight has been largely effected by increasing the weight of the head, the centre of gravity of the falling stamp being lowered thereby.

Of the falling weight the "shoe" alone comes in contact with the ore and suffers wear. It is therefore usually made of a hard steel, forged steel, chrome steel, manganese steel, etc., the first being the most common though probably forged chrome steel makes as good a material as any. Where mines have their own foundries chilled cast-iron may perhaps be advantageously employed. This shoe consists of a shank which enters the head, and a butt which bears the wear (Fig. 90). The diameter of the butt is the same as that of the head, while its length varies from 8 to 12 in. If longer than this there would be too great a difference in the falling weights of new and worn stamps. A shoe usually weighs 200—300 lb. and forms about 15 per cent of the stamp weight.

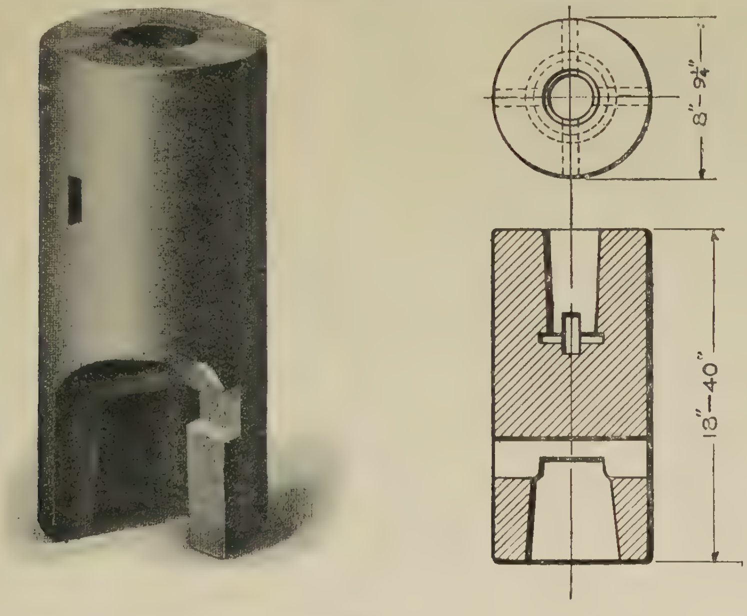


FIG. 89.

Head or Boss.—A cylindrical mass connecting stem with shoe, the former entering a bored socket at the top, and the shank of the latter entering a cast socket at the bottom. At the end of each socket is a driftway for the introduction of a drift to assist in detaching stem or shoe, these two driftways being at right angles. To have them in the same plane would weaken the head (p. 134).

The last item in the falling weight is the "tappet." This piece is a cylindrical casting bored to fit the stem, upon which, usually about two-thirds of the way up, it is fixed by gib and key (Fig. 91). At either end its face is machined so that it may be reversed upon the stem, the lower face being that which the cam engages. The weight of a tappet varies with the weight of the other falling parts, and is usually from 100 to 250 lb. A heavy stamp, requiring a longer gib and more keys, requires a heavy tappet.

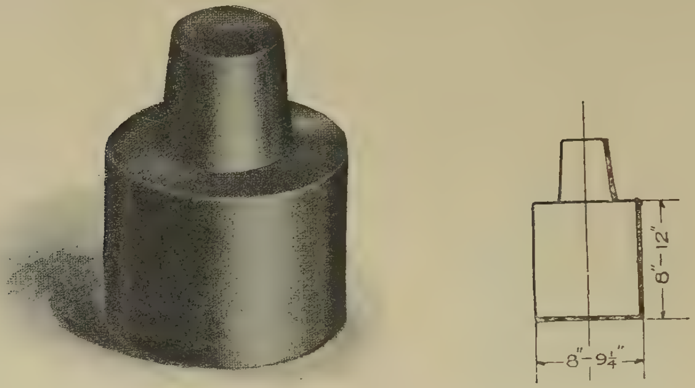


Fig. 90.

Shoe.—The wearing part of the falling stamp is the shoe. This piece consists of a shank which enters the head and a butt which suffers the wear, the shank being small and tapered while the butt has the full diameter of the head (p. 135).

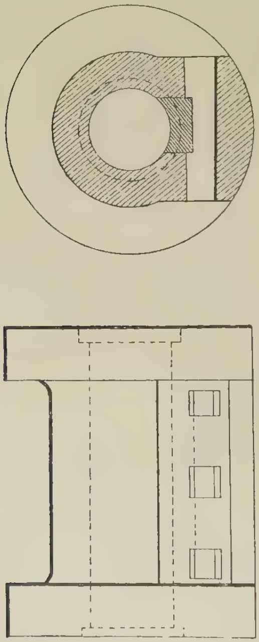


Fig. 91.

Tappet.—The piece by which the stamp is caught and lifted is the tappet. This piece is in the main cylindrical, bored axially to slip along the stem, and having flanges at either end to form the wearing faces. It is held fast to the stem by keys, which when driven home press a gib tight against the stem. Three keys are shown, this being normal, but heavy stamps require two gibs and four keys. It will be noted that each wearing face is counterbored around the stem to permit the face, under repeated blows, to spread inside as well as outside, the face being thereby maintained horizontal during wear (p. 135).

Putting all these different items together, the following total falling weights and percentages are obtained :

Stem	300— 600 lb. =	35 per cent.
Head	300— 650 lb. =	37·5 ,,
Shoe	150— 250 lb. =	15·5 ,,
Tappet	100— 200 lb. =	12·0 ,,
	850—1700 lb. =	100·0 per cent.

A falling weight of 1000 lb. when the shoe is new, may be taken to be about the average of present-day practice, and that of 2000 lb. as the extreme weight possible by cam-and-tappet lifting. At the Homestake mine, Dakota, where modern stamp-milling was largely developed, the falling weight is about 900 lb. and the diameter of head about $8\frac{1}{2}$ in.

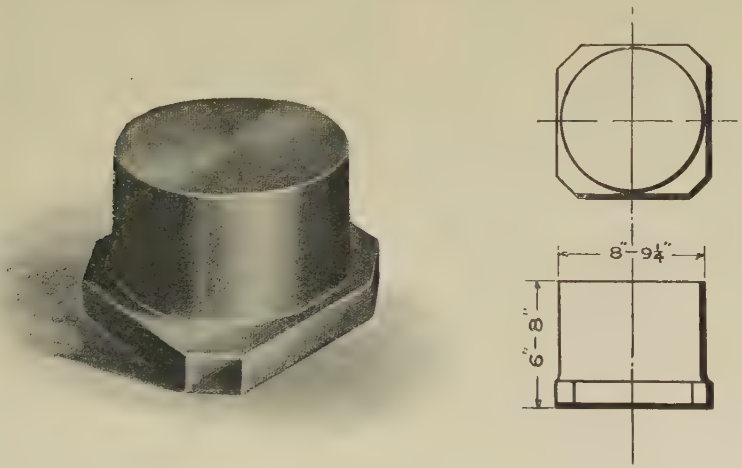


Fig. 92.

Die.—The die is the anvil on to which the stamp falls. It consists of a circular butt upon a square base, the corners of this base being removed to give purchase to lifting bars inserted when the die is to be changed. The butt has the same diameter as the shoe ; the base is of a size to fit into the bottom of the mortar box, possibly with liner plates between (p. 137).

Each stamp falls on a separate "die," which is circular and of the same diameter as the shoe, though its base is square the better to fit the mortar box (Figs. 92, 93). These dies are less deep than the shoe, since even when of the same material, being protected by a cushion of ore, they do not wear so quickly ; it is better to give shoe and die such respective dimensions that they may be discarded together ; a 9-in. shoe would generally require a 6-in. die. Often the die is made of softer material than the shoe, partly because the die does not suffer such a direct blow, but also because though the wear of softer material is greater it is more uniform.

Five of these stamps work together in the "mortar box," which is a heavy iron casting with a particularly heavy base, flanged front and back but not at the ends (Figs. 93-95). At the top this box is open, at the back is the feed opening, while in front is the discharge and a lip over which the crushed pulp flows. The weight of such a box varies somewhat with the weight of the stamp, but not proportionally, because the dimensions of a box are determined rather by the diameter of the stamp than by its weight,

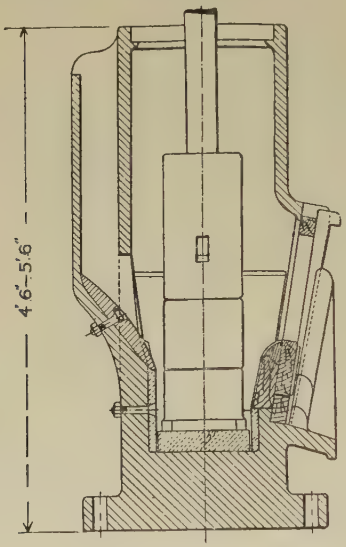


FIG. 93.

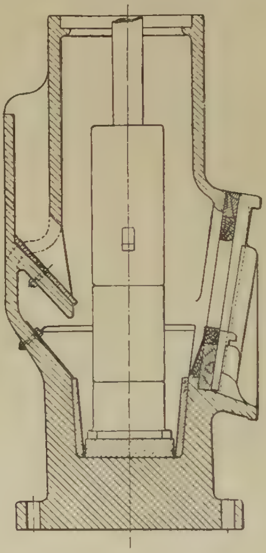


FIG. 94.

Mortar Box (Fig. 93).—Cross-section. This section shows stem, head, shoe, die, and false die, assembled within the mortar box, this box being protected by liner-plates along the side, at the ends, and in the feed entry at the back. The discharge opening in front is occupied by a wooden chuck-block below, and a screen stretched upon a wooden frame above. This frame and the chuck-block are held tight against the casting by two long steel keys, one at each end; two supplementary and shorter keys, again one at each end; and two wooden wedges buttressed against lugs cast upon the lip. In the illustration the long keys are shown and the lugs against which they take their purchase, but the smaller keys and the bottom wedges are only indicated by their lugs (pp. 137, 138, 142, 143).

Mortar Box for Inside Amalgamation (Fig. 94).—Cross-section. This illustration shows the recess at the back whereon an amalgamated copper-plate is carried; in contradistinction, the ordinary mortar box apart from the feed opening has a straight back. It will also be noted that the die sits deeper in the casting, so that the height of discharge may be made suitable to the finer crushing proper to inside amalgamation (pp. 138, 142).

and heavy stamps have much the same diameter as light stamps. The weight depends more upon the depth of metal in the base, which may be as much as 12 in. but is generally considerably less, especially where a removable false bottom is used inside the box. Separate external anvil blocks—tried as a means of giving the necessary solidity to a relatively light box—have not been continued in recent designs. The open removable front introduced for convenience in handling the long heads of heavy stamps, also reduces the weight of the casting (Fig. 96), as do also the rounded corners designed to lessen the clearance space around the end stamps. At the Homestake with 900 lb. stamps the box weight was 7000 lb.; elsewhere with 1750 lb. stamps it was 10,500 lb.; while in a third mill with 1400 lb. stamps it was 12,000 lb. A weight of 10,000 lb.

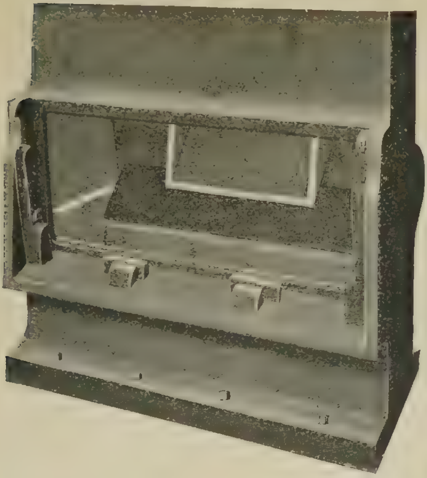


FIG. 95.

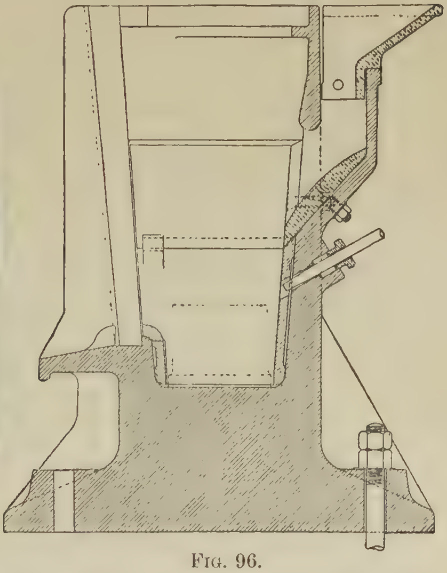


FIG. 96.

Mortar Box (Fig. 95).- Perspective Front View. In this view chuck-block and screen have been removed. Inside the box can be seen the feed opening, an end liner, and the back liner, as well as a plate for inside amalgamation. At the front, the keys, wedges, lugs, and discharge lip are well shown (pp. 138, 142).

Open-front Mortar Box (Fig. 96).—With this box the front, above the lip, is a movable piece, and not, as with the ordinary mortar, a part of the box casting. Such an open front facilitates the manipulation of the long heads of heavy stamps. In the illustration the water used in crushing is seen to be entered at the back and to be so directed that the stream strikes the surface of the die, the idea being that the material already crushed fine enough shall thereby be swept away and not remain to cushion the blow upon the coarser pieces. Ordinarily the water is entered through two pipes at the open top of the box from a service running along the front of the stamps (pp. 139, 142).

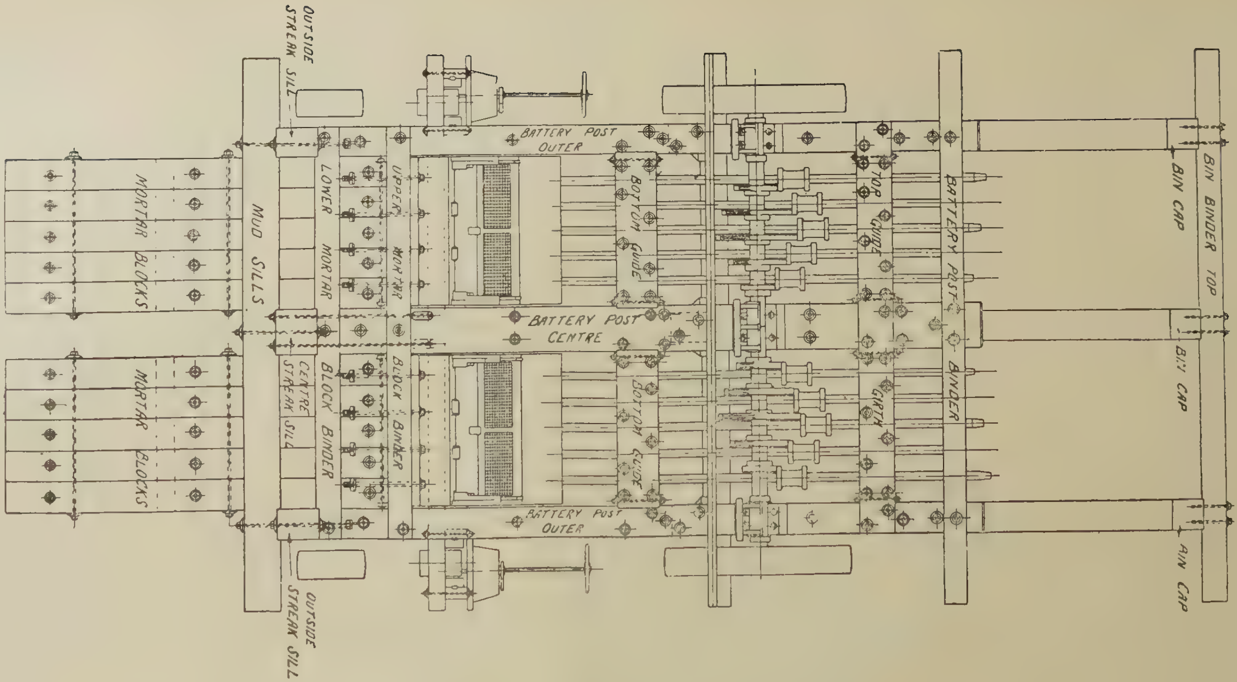


Fig. 97.

Stamp Framework.– Front Elevation. The mortar blocks and framework for ten stamps are shown. The binders which, yet without bolts, hold the blocks upright between the posts, deserve attention. The framework has the centre king-post common to both batteries of 5 stamps, and of double width suitable to that double service. At the top of the posts comes a binder which is continuous along the batteries, the posts thereby mutually supporting themselves against lateral thrust.

The left-hand five stamps have right-hand cams, and the right-hand five stamps left-hand cams, the two cam shafts thereby thrusting inwards against the centre post (pp. 143, 148, 149).

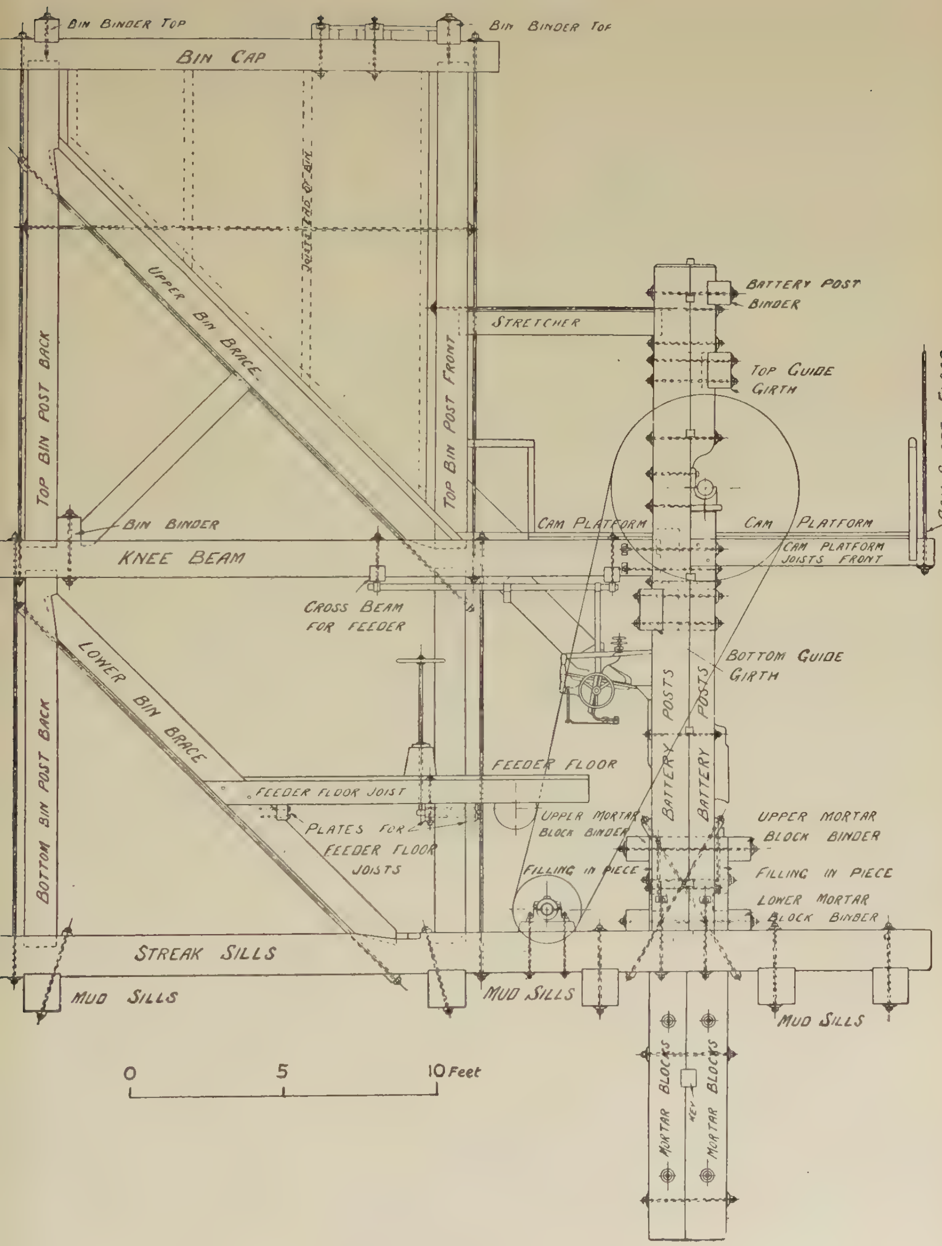


FIG. 98.

Stamp Framework.—Side Elevation. The particular illustration is of the "back-knee" framework, the knee beam extending behind. Wooden mortar blocks are shown, around which, but independent of them, the framework is raised. The mortar boxes alone are bolted to the blocks; the holding-down bolts are shorter than usual, their lower ends being well above the mud sills. The framework is well braced against the tendency of bin pressure to push the posts forward and out of the vertical, back tie rods assisting. It will be noted that the top guide-beam is in front of the posts and the lower guide-beam behind the posts. On the feeder floor the rack-and-pinion belt tightener, by means of which the driving belt is caused to grip or to be released, is shown; and above that floor the suspended feeder, automatically operated by the falling stamp through a special tappet and a rocker arm. The feeder is of the rotating disc type (pp. 143, 144, 155).

may therefore be considered to be a fair average weight with present-day stamps.

Concerning dimensions, the mortar box for an average weight of stamp is about 5 ft. in height and 4 ft. 9 in. in length. In width, that of the base is usually about 2 ft. 4 in., that internally at the surface of the die about 14 in., and that at the level of the discharge about 22 in. Practising inside amalgamation the internal width would be greater, to provide collecting room for the amalgam (Figs. 94, 95). A narrow width characterizes the box used solely for crushing.

The feed opening at the back extends over the middle half of the box and is wide enough to take the coarsest piece likely to arrive, namely, about 3½ in. Falling down it, the ore drops on to a slope directing it under the middle stamps, whence by the sequence of drops it is distributed to the end stamps. Water is fed into the box either by way of the feed opening, or from the open top, or down ways so placed in the back that a stream falls on to each die (Fig. 96).

Discharge is effected through a screen of woven wire or punched plate fixed on a wooden frame fitting the discharge opening (Figs. 93-95). This screen frame, with a strip of blanket around it, is held against the box at an angle of about 10° from the vertical, by steel keys at either end and by wooden wedges on the lip. Against the screen the pulp at each drop is splashed, that fine enough finding exit. In order that discharge should take place freely, the screen should be kept as near to the stamp as possible, or otherwise only the finest particles would reach it. The width of a box has therefore a pronounced influence upon the rapidity of discharge, and consequently upon stamp duty.

The height of the level of discharge above the die is also an important factor, the greater this height the finer the discharge. So important is the "height of discharge" that though the aperture of the screen may determine the maximum size of the product and prevent large pieces from passing, the character and the quantity of the product are largely determined by the height above the die at which discharge takes place. This height varies according to the desired product from 2 to 15 in., the lower figure connoting a quick discharge and a granular product, the latter figure a delayed discharge and much slime. Whatever the policy followed, the height of discharge should be kept fairly constant as the die wears, which may be done by seating the screen frame not directly down upon the lip of the casting, but up upon an interposed piece known as the "chuck-block" which, as wear proceeds, may be changed for others of smaller dimension or be taken out altogether (Fig. 93).

The mortar box sits upon what is termed a "mortar block." This block was long constructed of wood. It was feared that concrete would not stand the disintegrating pounding of the stamps, nor the stamps the shivering rebound from so inelastic a material. But these fears have since been proved unfounded and blocks are now made preferably of concrete.

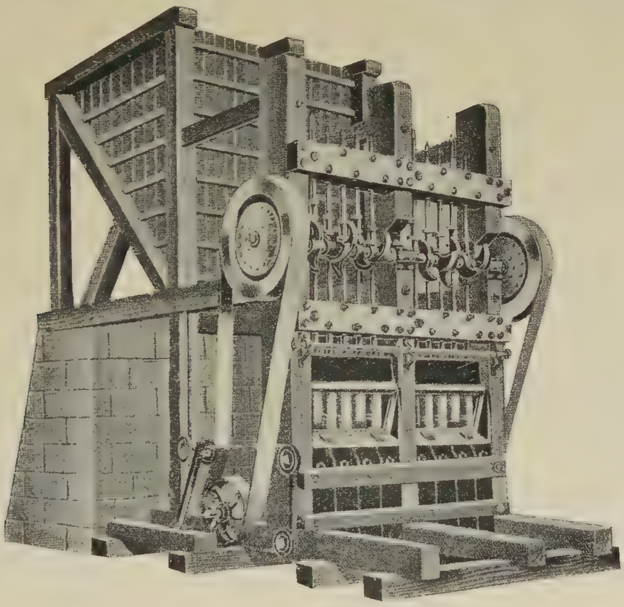


FIG. 99.

General View of a Stamp Mill (p. 144).

Dealing first with the wooden blocks, each block is a little bigger all round than the mortar base, and of sufficient depth, about 12 ft., to absorb the blow (Figs. 97, 98). Such a block is built of separate timbers bolted and braced together, sometimes even of planks spiked together. It is set on concrete or other good foundation, in a pit of such depth that about 4 ft. of the block projects above the ground, and to it the mortar box is held by eight heavy bolts, of $1\frac{1}{2}$ in. or 2 in. iron, four on each side. These 'holding-down' bolts extend down into the block to about ground level, where they are held by cotter and plate inserted from the outside, a method of holding which permits a bolt to be released and withdrawn in case of breakage. Whatever the design or the material of the block, this withdrawal should be possible.

In dry climates such wooden blocks last a long time, but in hot, moist climates the best of timber has a short life under the alternate wetness and dryness in which that portion above ground finds itself, and renewal and repair, particularly around the bolt ends, become expensive items. A normal life would be twenty years under good conditions, but less where the conditions were unfavourable.

Concrete mortar blocks are made of an ordinary concrete-mixture, such, for instance, as cement 1, sand 2, and stone 4, with perhaps less cement in the lower portions and a little more cement in the upper portions, to give the block greater strength immediately under the blow. In shape such a block is wide at the bottom and narrow at the top (Fig. 100). Down the sloping sides the holding-down bolts are laid in slots, taking their purchase from straining blocks in pockets about half-way down the slope. With bolts so laid the box requires a somewhat wider base than usual, together with special seatings or angle-washers. The depth of such a block is about 8 ft., which is also about its bottom width. With such dimensions the block is a heavy monolith, in most cases unnecessarily heavy.

The weight and consequently the cost of a block can be reduced without interfering with its strength, by leaving in it a central tunnel from end to end, a tunnel large enough for a man to enter to attend to the holding-down bolts which then are arranged to reach it (Fig. 101).

On top of the block and between it and the mortar box is laid a thickness of sheet lead, rubber insertion, belting, or tarred blankets, in which under the blow the machined bottom of the box may bed itself, and upon which the box may be tightened without risk of springing the flange. A quarter of an inch of such material is sufficient if the upper surface of the block has been well formed (Fig. 101).

Concrete blocks, as inferred above, are now the common construction. They are cheaper to set up, require neither renewal nor repair, and apparently render the blow more effective. Since also they neither shake nor settle unevenly under the blow, the foundations for the upper framework may be made in one piece with them, a practice never followed with wooden blocks.

Where wooden blocks are used, this upper framework, carrying the driving, feeding, and guiding arrangements, rises from large timber sills running forth and back between the boxes (Figs. 98, 99). These sills, known as 'streak sills,' are in turn supported by 'mud sills' running at right angles, that is, along the line of boxes, and upon the ground. Upon this platform of sills the posts are erected. Principal among these are the 'king-posts' which stand upon the streak sills, to which in turn they are firmly bolted. On either side of each box is such a king-post, 16—20 ft. in height. The centre king-post between each two boxes is generally common to both,

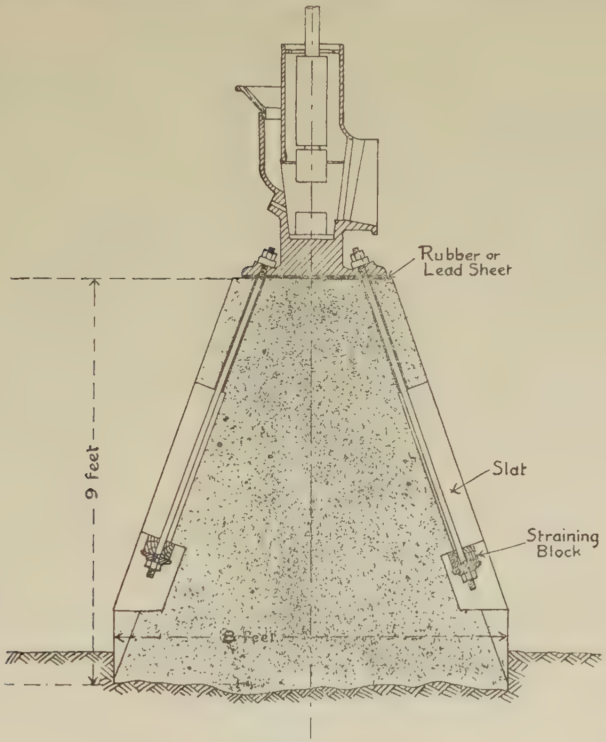


Fig. 100.

Concrete Mortar Block.—These blocks are not so deep as wooden blocks, but expanding downward they have a wide base. In the illustration a width of 8 feet at the base is shown and a height of 9 feet, such dimensions being only warranted with very heavy stamps. The holding-down bolts lie, as will be noted, on the sloping flanks of the block, passing actually through concrete in their upper portions and lying exposed in open slots in their lower portions. These bolts, which are very long, take their purchase against wooden straining-blocks below and against special seats on the flanged base above. The open slots and the clearance space below permit these bolts to be withdrawn downwards (p. 144).

the outside posts then marking off ten stamps. Notched and bolted to these posts and extending behind them are horizontal 'knee-beams' which support the 'ore-bin,' these beams being themselves supported by posts standing about 5 ft. behind the king-posts, and by other posts or by masonry still farther behind at the back of the ore-bin. Above the

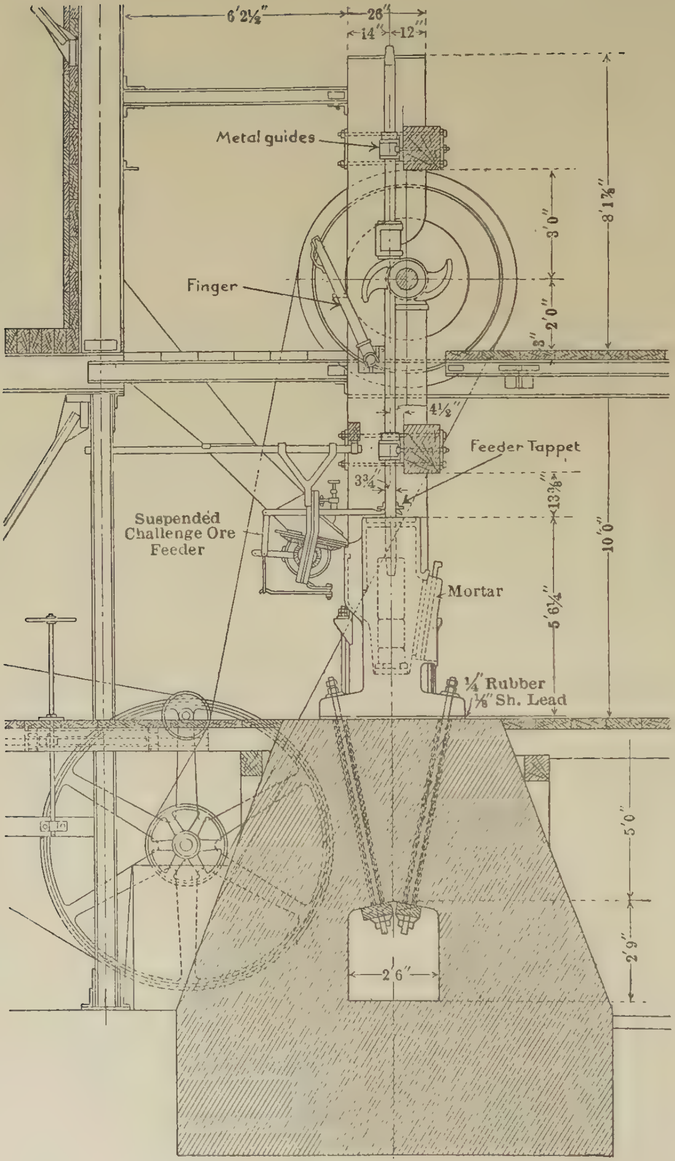


FIG. 101.

Concrete Blocks and Steel Ore-bins.—This illustration shows a concrete mortar block with a central tunnel from end to end into which the holding-down bolts are brought. This tunnel is large enough for a man's entry, and it has such clearances (not shown) in its floor that the bolts may be withdrawn ; it reduces the amount of concrete otherwise necessary, without weakening the block. The block shown is, however, of very generous dimensions, particularly as the stamp weight was only 1250 lb.

Other interesting points in this illustration are: the fingers upon which the stamps may be supported, or hung up, when not working; the metal guides for the stamp stems; the feeder tappet which strikes the rocker arm of the feeder; and the belt tightener (*Trans. A.I.M.E.* vol. lii. p. 95) (pp. 144, 147, 155).

knee-beam, and in line with the front support, are other posts forming the face of the ore-bin, these upper posts being strutted against the king-posts by distance-pieces or stretchers. To keep the king-posts from being pushed forward by pressure from the ore-bin, tie-rods are placed along the ore-bin braces, the distance-pieces, and the posts, the whole structure being thereby anchored to the back.

With concrete blocks the upper structure is much the same as with wooden blocks, but the platform of sills is modified and may even be cut out altogether (Fig. 101). Sometimes, indeed, the concrete encroaches upon the upper framework, cutting the king-posts short, but this departure has met with little approval.

In other designs the whole framework, including the posts, is of steel or wrought iron, but experience with such frames has been less satisfactory than with wooden structures, largely because the riveted connections do not so well withstand the continual vibration, nor are the nuts so easily kept tight.

Dealing now in particular with the guiding arrangements, stretching across the front and holding the posts of ten stamps together, are two 'guide-beams,' one below and the other well above the knee-beam. Attached to the back of these guide-beams are the "guides" which hold the stems vertical throughout their movement and keep them centred over their respective dies. These guides may either be solid, that is, with all five stamps passing through the same piece, or sectional, with each stem passing through its own separate guide. When solid, they consist of two pieces of a hard, fine-grained wood, one placed along the back of the stems, and the other along the front, both having semicircular ways so cut in them that when the two halves are put together five circular passages for the stems become complete (Fig. 102). These two halves are drawn together by bolts extending through the guide-beam, and kept from too tightly embracing the stem by wedges driven between the two halves.

Sectional guides are generally metal sleeves, two halves lined with soft metal being brought together to embrace each stem; they may, however, also be made of wood. In either case the actual guide-pieces are held in iron brackets bolted to the guide-beams (Fig. 101).

With respect to the manner and means of lifting, these stamps, like Cornish stamps, are lifted by cams. In the place of the hollow cam-barrel, however, there are cam hubs upon a "cam shaft" (Figs. 97, 103). This cam shaft is ordinarily about $5\frac{1}{2}$ in. diameter, and of hammered malleable iron or mild steel, harder steel breaking under the repeated blows. Its length depends upon the number of stamps worked from it. With small batteries and batteries of medium size this number generally is five. In that case, the two short shafts of a ten-stamp unit have separate bearings upon the centre king-post, each shaft having its own driving-pulley overhanging an outside post. But in larger batteries ten cams are usually driven together to avoid an inconvenient multiplicity of belts. The cam shaft then has three bearings, one on each of the outside posts and one on the centre post. There is, however, this disadvantage when driving

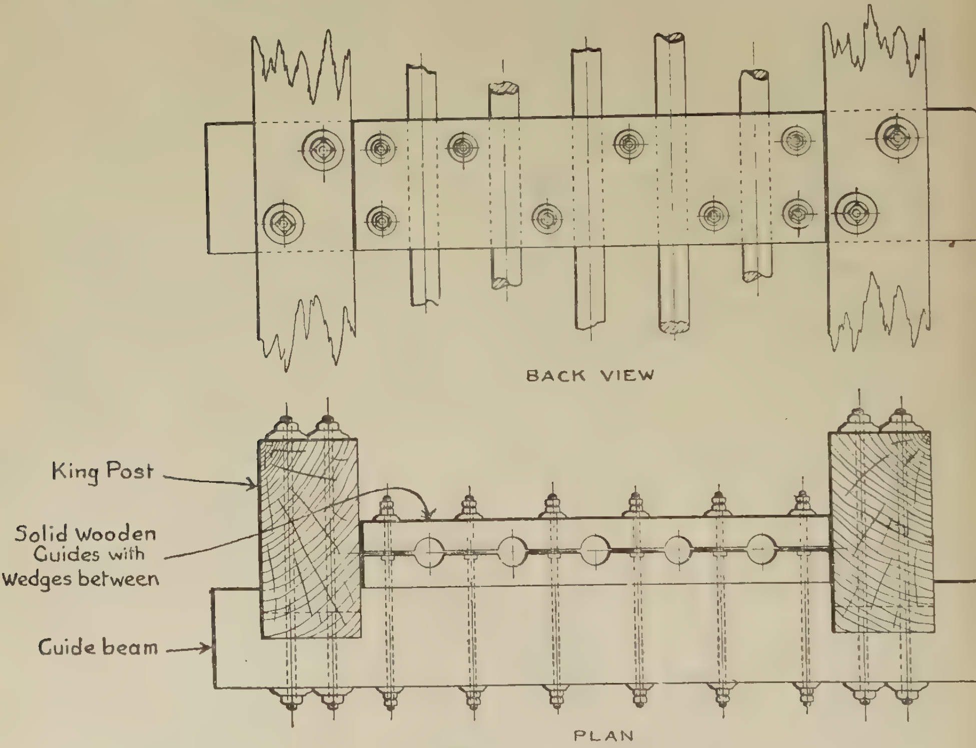


FIG. 102.

Guides.—The solid wooden guides are illustrated. Two deep planks are so grooved, each on one side, that when the two are placed alongside, five circular holes remain between them for the passage of the stems. Placed at the back of the guide-beam, the inner plank or half-guide is held tight against the beam by a collar on a through-bolt, while the outer half is brought up by a nut on the end of the same bolt. Wedges driven down between the two halves maintain the distance necessary to the easy passage of the stems (p. 147).

ten together, that if a cam or cam shaft breaks more stamps are hung-up while repair is being effected.

The cam-shaft bearings are generally lined with soft metal though sometimes the steel shaft bears directly on cast-iron. In either case, the speed being low and the pressure on the bearing great, a stiff grease is necessary for lubrication. These bearings not only support the cam shaft, but with collars on that shaft they prevent its lateral movement. They themselves are fixed on seats cut on the front of the king-posts just above the knee-beam. A platform laid on those beams accordingly serves as a cam-shaft platform, convenient for attention to all the lifting parts.

The "cams" themselves are not simple tongues, but arms springing from the hub, these arms having an involute face backed by a web (Fig. 103). Usually, two such arms, set at 180° to one another, have a common hub bored to slip over the cam shaft, the whole system of arms, web, and hub, constituting the cam. This cam is a steel casting. The arms do not spring medially from the hub but from the end adjacent to the stem to be lifted (Fig. 104). Away from that stem the hub extends half-way towards the next cam. Since a tappet may be engaged from either side of the stem, this extension of the hub may be either to the right or to the left of the arms. When, looking in the direction the arm is moving during lift, the hub extends to the right, the cam is a right-hand cam, and *vice versa* (Fig. 104). The stem axis not being in the plane of the cam revolution but to one side, during lift there is a tendency for the cam to be forced away from the stem, a lateral thrust resulting. This thrust may be met by having right-hand cams on one side of the centre post of a ten-stamp unit and left-hand cams on the other, the two lateral thrusts then largely annulling one another, to the relief of the frame (Fig. 97). At the same time undue eccentricity of lift is avoided by bringing the cam as close as possible to the stem, the clearance between hub and stem usually being about $\frac{1}{4}$ in.: in that position the eccentricity remains sufficient to rotate the stamp, the shoe then wearing evenly (Fig. 104).

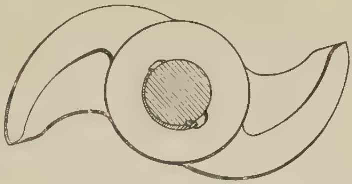


FIG. 103.

Cam.—A two-armed cam, with hub, web, involute flanges, cam shaft, and tightening wedge, is illustrated. It will be noted that the involute face gives place at its end to a short circular length, the uniform lift of the involute giving place at the same time to the diminishing lift of a circular movement (pp. 149, 153).

As will be realized, the tappet being circular and placed co-axially upon the stem, overlaps the line of lift and to that extent stands directly over the hub (Fig. 104). It is obvious, therefore, that the position of the

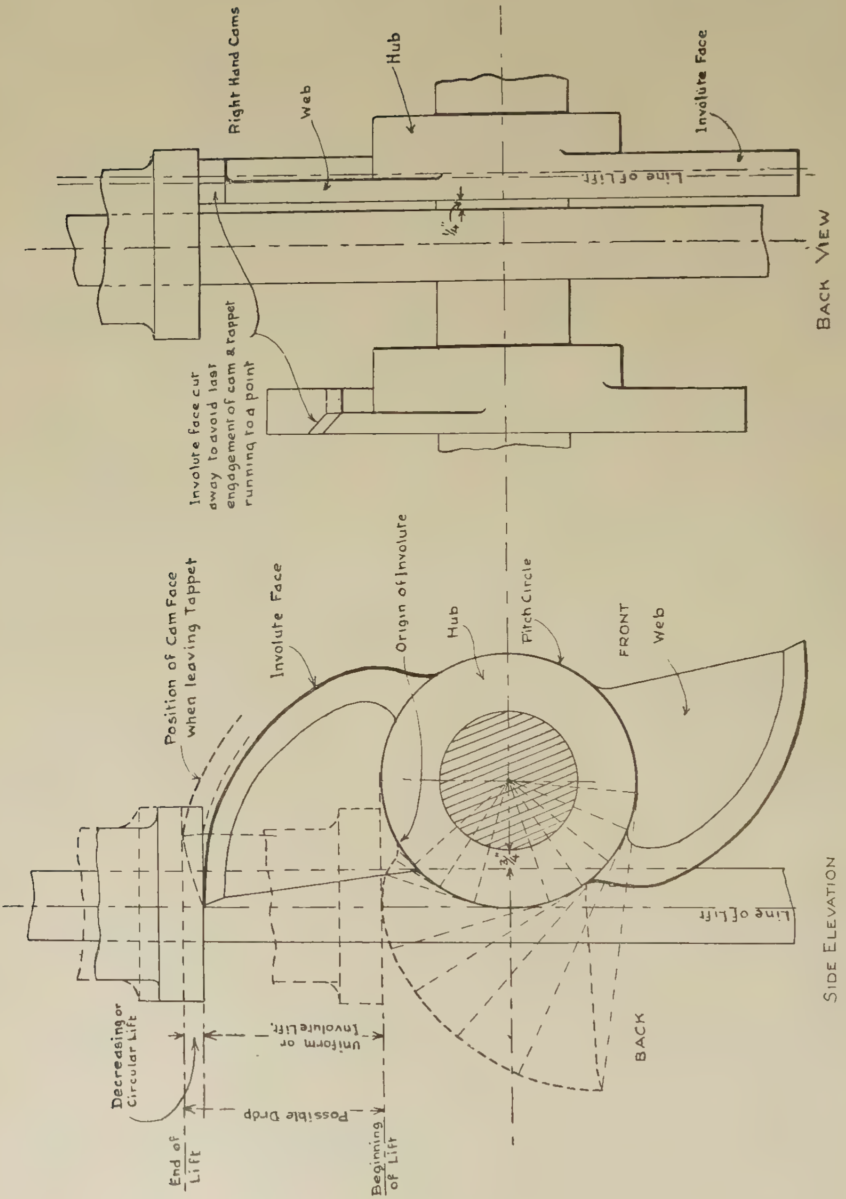

Right Hand Cams
Web
Hub
Involute Face
Line of Lift
¼"
Back View
Involute face cut away to avoid last engagement of cam & tappet running to a point
Position of Cam Face when leaving Tappet
Involute Face
Origin of Involute
Hub
Pitch Circle
FRONT
Web
¾"
Line of Lift
Side Elevation
BACK
Decreasing or Circular Lift
End of Lift
Uniform or Involute Lift.
Possible Drop
Beginning of Lift

FIG. 104.

Diagram illustrating the Lifting of Californian Stamps (Fig. 104).—In the side elevation the tappet is shown possible of drop to the level of the top of the cam hub, from which level, therefore, lift would begin. In practice, drop would probably stop about an inch above that level in order to secure an unquestionably sufficient clearance between tappet and hub. Yet with care the drop illustrated would give sufficient clearance. It certainly would give greater clearance than the drawing appears to indicate, since the tappet face instead of reaching straight out from the stem towards the hub curves inward, by reason of its circular section, and recedes from that hub; the true clearance is about that indicated over the origin of the involute.

With the drop envisaged in the illustration the cam face has to be so laid off, that, firstly, it presents itself for the beginning of lift as though it had already lifted the tappet from the plane through the centre of the pitch-circle; and, secondly, that it may continue lifting till the desired height of drop is obtained. The greater part of this lift is made by the regular progression of the involute face across the line of lift, but when the cam extremity crosses that line, lift no longer proceeds at the uniform rate which the involute gives, but at the diminishing rate of the circular movement of the cam extremity around the centre of rotation. Finally, the cam escapes from under the tappet, this escape being somewhat hastened by cutting away the near corner of the cam extremity.

This question of clearance only arises because the cam shaft is brought as close as possible to the line of stems—it will be noticed that the clearance between these pieces is only three-quarters of an inch. The cam shaft could, of course, be placed far enough away that the tappet in its fall could not possibly strike the hub; that indeed is the position with Cornish stamps, but the arms would then have to be much longer, and they would be relatively weak under the blow they unavoidably receive when picking up the stamp.

The back view shows the lateral relations of cam and stamp. Right-hand cams are illustrated, those with hub extending to the right when the cam is viewed in the direction the arm is moving as it lifts. The clearance between the stem of the stamp being lifted and the cam lifting it, is seen to be small, and in the illustration one-quarter of an inch. The line of lift is taken to be in the plane passing medially through the cam face (pp. 149, 152).

tappet on the stem must be so arranged that, in falling, the overlapping portion of the tappet shall not strike the hub; in practice this possibility is excluded by not permitting the tappet to fall so far.

From the above-described disposition of tappet and hub it follows that the lift of the tappet does not begin at the origin of the involute cam, but at that point along it where the pitch-circle tangent has the length of the hub radius. In that way vertical clearance between hub and tappet is provided, and the cam shaft may be brought so close up to the line of stems that between the two there is only about half an inch of clearance. Accordingly, in determining the necessary length of cam face it must be assumed that the total length of lift is equal to the desired lift plus a length equal to the hub radius. It must also be remembered that though, during lift, the point of contact between the involute and the tappet face describes a vertical line—the line of lift at a uniform velocity—at the end of that regular engagement the extremity of the cam has still to get out from under the tappet; and, that it accomplishes this escape by moving in a circle around the cam shaft, lifting the tappet at a decreasing rate until the two are free (Fig. 104). Accordingly the involute face is not quite responsible for all the lift.

The involute of the cam face is often drawn to a pitch-circle coincident with the hub circumference, such an involute giving a suitable velocity of lift. Whether or not this circumference would be the natural or normal pitch-circle would depend upon the position of the cam shaft. If that shaft were very near the stamps and the hub were a large one it would be greater than the natural pitch-circle; and, conversely, if that shaft were far removed it would be smaller.

The arrangement of the cams around the cam shaft is no less important than their relative positions along that shaft, since the order of drop is thereby determined. The five stamps cannot all be lifted together, the stress would quickly break the shaft. Nor can they be dropped in succession one after the other without forcing too much of the ore towards one end or other of the mortar box; to accomplish effectively their part in distributing the feed no two adjoining stamps should fall consecutively.

Several sequences of drop have been found satisfactory. First of all it must be said that standing in front of a battery, the stamps are numbered from left to right, 1, 2, 3, 4, 5. Viewed from the back and numbered from left to right again, the same succession would be represented by 5, 4, 3, 2, 1, the reciprocal of the previous sequence. Similarly, for every order of drop there will be a reciprocal order having the same distributive effect, except that if the original order tends to drive the material towards one end of the box the reciprocal order will tend to drive it towards the opposite end.

In California the order 1, 4, 2, 3, 5, 1, and its reciprocal 1, 5, 3, 2, 4, 1; the order 1, 5, 2, 4, 3, 1, and its reciprocal 1, 3, 4, 2, 5, 1, are used; and in South Africa 1, 3, 5, 2, 4, 1, and its reciprocal 1, 4, 2, 5, 3, 1. Of these the Californian sequences give a more uniform distribution of the feed, the African sequence sending the material a little towards one end of the box, necessitating an increase in the drop of the stamp at that end. If ten stamps are run upon one shaft it is usual to interpolate the second five between the others so that the ten lifts are regularly distributed around the circumference.

Whatever the sequence, it is obtained either by having one long key-way cut along the shaft, the key-way of each separate cam being cut to suit its position on the shaft; or the cams are fastened by some sort of wedge. The Blanton wedge, representative of the latter type, is a wedge enveloping an arc of about 180° on the shaft, and having its thick end pinned to the shaft, the shaft being bored to receive the pins in proper place for the desired order of drop (Fig. 105). This wedge has its thin edge forward in the direction of rotation, while in the hub of the cam is a similarly-shaped recess (Fig. 103); accordingly, in rotation the blow against the tappet tends to tighten the cam still further, whereas reverse blows with the hammer will effect its release.

The speed at which stamps are run largely depends upon the height of drop considered proper, having regard both to the material to be crushed and to the stamp weight. With a low drop of $6\frac{1}{2}$ in., suitable to a relatively soft ore, cams of normal involute shape will allow a speed of 110 drops per minute, whereas with a high drop of $9\frac{1}{2}$ in., suitable to a hard ore, 90 drops could hardly be exceeded. Where complete pulverization by stamps is desired, as for amalgamation, the height may be still greater; in places and in times past, for instance, a drop of about 15 in. was usual, and a number as low as 40 per minute.

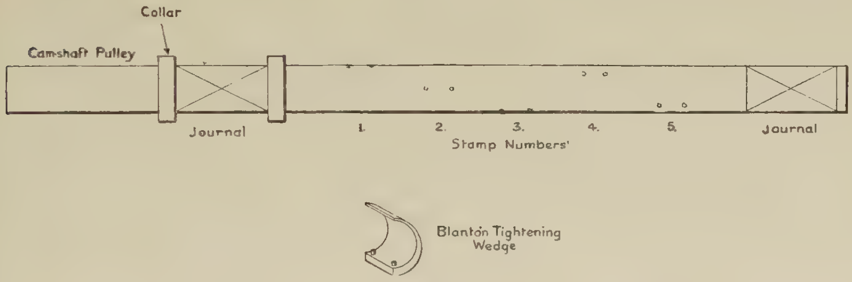


Fig. 105.

Cam Shaft bored for Tightening Wedge.—The cam shaft illustrated is for five stamps; the journals rotate in bearings on the outer and centre king-posts respectively, the cam-shaft pulley overhanging the former. The shaft is bored for the stamps to drop in the order 1, 4, 2, 5, 3 (p. 153).

The time taken for each complete blow includes that occupied in lift and that in fall. In addition, in order that the stamp shall always fall completely and without arrest by reason of the tappet striking the next oncoming arm—a hurtful occurrence known as "camming"—it includes also a clearance interval between the end of fall and the beginning of the next lift. The magnitude conceded to this interval will depend upon the variations in speed of the prime mover by which the stamps are driven; it is generally about a fifth of a second, the stamp during this time making its rebound. Being fixed independently of the number of drops per minute, the greater this number the greater the total clearance time. These points are well brought out by a graph (Fig. 106), but are also contained in the following figures of number and height of drop sanctioned by experience:

Number of Drops per Minute.	Height of Drop.	Total Lift per Minute.
110	6½ inches	715 inches
100	8 „	800 „
90	9½ „	855 „

It might appear from the above figures that the capacity of a given stamp would be greatest with low speed and high drop. But that is not so; the height of drop should not be greater than that necessary to give the desired blow, and at that height the stamp should be run as fast as possible. The capacity then depends upon the number of blows made in a given time, a number determined by the time taken for each blow. To increase that number the time taken for each blow must be diminished. But the time occupied in fall cannot be diminished, nor with safety can the clearance interval. Accordingly, the only remaining chance to increase the number of drops is to shorten the time of lift. Increased speed of lift can be obtained by making the cam face an involute to a pitch-circle greater than the normal.

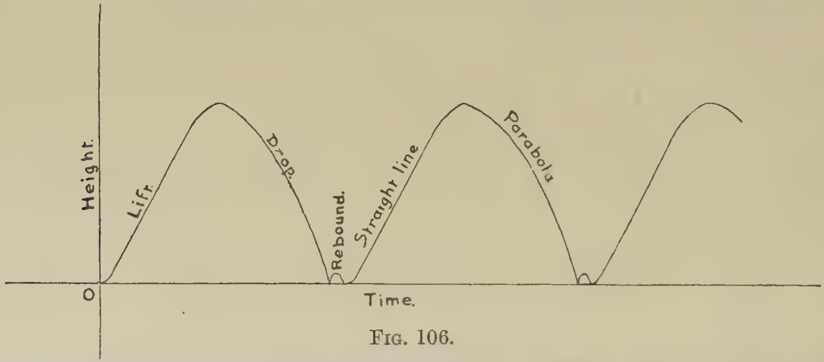


Fig. 106.

Graph of Lift, Drop, and Rebound.—Involute lift when plotted between co-ordinates of height and time presents the straight line of uniform movement. At the top, as the cam extremity moves out from under the tappet, the lift is due to circular movement and the curve flattens. Drop is at the increasing speed which gives a parabolic curve. Rebound takes place in the clearance interval between the end of drop and the beginning of the next lift. At 90 drops per minute the whole sequence of lift, drop, and rebound would occupy one and a half seconds (p. 154).

The normal pitch-circle of the involute cam has for centre the axis of the cam shaft and for radius the horizontal distance between that axis and the plane containing the stem axes. The involute to such a circle would lift at the circumferential speed of that circle. By constructing the cam to a pitch-circle of greater radius the speed of lift is proportionally increased, and the permissible number of drops per minute increased somewhat. Under these abnormal conditions, however, the tangent to the cam surface at the point of contact with the tappet is no longer horizontal, and lift no longer quite vertical, thrust resulting. A larger yet normal pitch-circle may also be obtained by opening-out the natural radius, that is, by setting the cam shaft further from the stamps ; but with the greater length of arm thereby rendered necessary the cam would have to be more than proportionately heavier ; moreover, the cam shaft would not then be so well accommodated upon the king-posts of the battery framework.

The number of drops per minute possible with cam lifting, though dependent primarily upon the height of drop, therefore depends also upon the curve of the cam. In practice this curve approaches the normal involute and 'quick-lift' cams are the exception.

The height of drop is measured down to the die. In operation, however, there is always a depth of ore upon the die, this depth depending upon whether the ore is fed heavily or lightly. A heavy feed lessens the number of breakages, but cushions the blow ; a light feed increases the stamp duty, but, the stamp then occasionally falling actually to the die, it increases the breakages of the stem or other falling part. Usually the depth upon the die is about 1 in., this being the condition which best balances the advantages of safe running and the disadvantages of striking a cushioned blow. To keep an even feed an automatic feeder is necessary, this being usually of the disc type actuated from a special tappet on the middle stamp (Figs. 98, 101). The stamp then helps itself to more ore when the amount on the die becomes below the designed minimum.

The ordinary size of the material fed to stamp batteries is such as will pass a 2-in. ring, with heavy stamps perhaps a 3-in. ring. If coarser than this, the stamp is doing work which could be done more cheaply by the rock-breaker. If finer, there does not appear to be much advantage, the amount crushed per stamp not being proportionately increased ; in fact, if sand were fed into a mortar box it would be almost as difficult to effect its discharge as it would to crush and discharge coarse ore. Tests in South Africa to determine whether any advantage were gained by breaking ore to ½ in. before sending it to the stamps, showed no increase of duty. The effectiveness of a stamp is at its highest when falling upon pieces it is just capable of crushing, all its energy being then applied to disintegration by impact. But when falling upon small material much of the blow is expended in rearranging the separate pieces, and in the attrition which accompanies this rearrangement.

The size thought fit for discharge depends upon whether the stamp is used for complete comminution or is followed by grinding machines. Complete comminution is usual when amalgamation is the only or the principal subsequent treatment, the screen being then about 30—40 mesh. It is also practised in dressing such fine-grained tin ores as require crushing through 20—30 mesh. Where, however, grinding follows, a coarse screen is used on the stamps, the actual size of the aperture depending upon the capacity of the grinding plant and the desired fineness of the end product. If some of the material can be satisfactorily treated in the condition of sand, the product through 3 mesh or even 2 mesh may not be too large to deliver to the grinding machines, but where all has to be reduced to slime, more work is usually put on the stamp and a 12—20-mesh screen is used. Whether fine or coarse screens be employed, the ultimate fineness of the product depends, as already stated, to a large extent upon the height of the discharge. It might well happen that with a greater height of discharge the product through a 30-mesh screen would have no larger average diameter than that passing 80 mesh with a shallower discharge ; it would certainly, however, be less uniform in size.

Finally, the amount of water used influences the rate at which the material is discharged and consequently its granularity. Much water will carry the material away quicker and leave it more granular ; with little water, discharge is delayed and the product will contain much fine material. The amount of water varies from 4 to 14 tons per ton crushed. A clayey ore naturally requires more water than a clean ore. Ore being crushed under conditions of high stamp duty will require less water than under low duty, since the same amount of water passing through the mortar box will carry away the greater tonnage.

The power consumed by a stamp is measured by its weight, its height of drop, and the number of drops, plus 25—30 per cent for friction. In consuming this power, the stamp duty depends upon the work the stamp is set to do, whether, for instance, that be : complete crushing ; coarse crushing with regrinding for sand and slime ; or stage crushing with regrinding for slime only. Under these different policies the respective results might be as follows :

	Coarse Crushing for Regrinding to		Complete Crushing.
	Sand and Slime.	Slime.	
Screen mesh	3	12	30
Height of discharge	3 inches	6 inches	9 inches
Duty per stamp per 24 hours	10 tons	6 tons	4 tons
Power required	3½ h.p.	2½ h.p.	2½ h.p.
Weight crushed per h.p. hour	350 lb.	200 lb.	125 lb.
Cost per ton crushed	12d.	20d.	36d.

With respect to the wear of the two crushing surfaces, that of the shoe is regular, a good surface being maintained, while that of the die is irregular, the worn surface often having a depression or cup at the centre. This irregularity of wear is all the more pronounced if the stamp be not truly centred over the die, a ridge then forming on one side, against which ridge the stamp strikes a glancing blow, dangerous to itself and of diminished effect in crushing. Dies also wear most at the back where the feed enters, and when so worn they are sometimes taken out and reversed.

The gross consumption of steel in shoe and die is about 1 lb. of average steel per ton crushed, the shoe contributing somewhat more than the die. When discarded, the latter still retains about half its weight, while of the shoe only about 20 per cent remains. Forging renders shoes and dies denser and more uniform in quality.

Though by far the greatest portion of the wear is borne by the crushing surfaces, the sides of the boxes also suffer. To protect these, liner plates are bolted or fitted to the back and sides, the wear on the back plate being sometimes considerable.

Heavy stamps have naturally greater capacity, greater roughly in proportion to their greater weight. Crushing the same material a heavy stamp could be run with a lower drop and, consequently, a greater speed; capacity would thereby be increased though, on account of the greater sum of the clearance intervals, not proportionally to the increased weight. The other alternative with heavy stamps is, while maintaining the same drop and speed, to obtain increase of capacity by feeding large-sized material, the heavier stamp then dealing with a greater depth of ore upon the die. This latter alternative is the more employed. Increase of capacity from the feeding of coarser material cannot, however, go on indefinitely, since material above about 3-in. ring is better reduced by a breaker; in the end also, upon a die-area increased but very little, the blow of increased weight becomes needlessly strong, while interference may also come from the cushioning effect of its own products. To avoid such cushioning a quick discharge is necessary, and, accordingly, the high duty associated with heavy stamps is due in more ways than one to the coarseness of the screen employed.

With their increase in capacity, heavy stamps have not, however, shown any greater efficiency in the use of power. Their advantage rather lies in economy of plant, shafting, belts, housing, etc.

Nissen Stamp.—The Nissen stamp works in a separate mortar box circular in section, and the difficulty of casting a large mortar box does not arise. In consequence, it has been possible to increase the stamp diameter to 10 in., and, in addition, to apply a semicircular screen concentric with the stamp (Fig. 107). The larger diameter, in conjunction with a long head of even greater diameter in its upper portions and a short thick stem, keeps the centre of gravity low and diminishes the height of the framework; the concentric screen is more effective than a straight screen in receiving the radial splash from the stamp, and discharge is in consequence facilitated, an advantage heightened by the considerably greater screening surface which each stamp possesses (Fig. 108). Moreover, a circular box is again better than a rectangular box, in that there is less useless clearance space and no corners, so that as the head is lifted the material falls back upon the die, from which position it cannot untimely be removed by the fall of any adjacent stamp. Each stamp has its own feeder and does not wait upon any uncertain distribution. Otherwise the Nissen stamp is much the same as the Californian stamp. It is lifted by cam and drops by gravity. The essential parts of the one exist in the other also, and considerations applicable to the one apply more or less to the other.

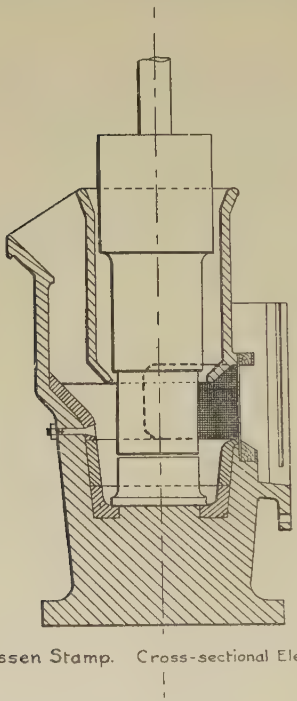


FIG. 107.

Nissen Stamp.—Cross-sectional Elevation. One stamp in its own box is the unit of a Nissen-stamp battery. This box is cylindrical, the stamp falling axially; in vertical cross-section, the feed opening, discharge, etc., are disposed much as with the ordinary box. The stamp head is seen to be of somewhat larger diameter than the shoe, and of still larger diameter in its upper portion. Shoe and die are of the same diameter, about ten inches (p. 158).

The Nissen stamp is essentially a heavy stamp, its weight being usually about 1600 lb.; it is at the same time a coarse-crushing stamp, since no fine screen would withstand so close a blow. Being such, its particular usefulness is to crush the breaker product in preparation for regrinding, generally by tube-mills. The height of discharge consequently is low and not more than about 2—3 in. The number of drops and height of drop are normal. When crushing to pass a 3—4-mesh screen each stamp of the above weight has a capacity of about 24 tons per day, equivalent to about 500 lb. per h.p. hour, the size-reduction being 20. Crushing to 20 mesh the capacity would be about 10 tons per day, and the size-reduction about 80. These results are about 25 per cent better than Californian stamps of the same weight would give, the equivalent saving in power consumption being about 20 per cent.

In respect to space occupied, two Nissen stamps take the place of five Californian stamps, that is to say, stamps of this design occupy somewhat more space per ton crushed than Californian stamps doing the same work. On the other hand, they do not require so high a framework.

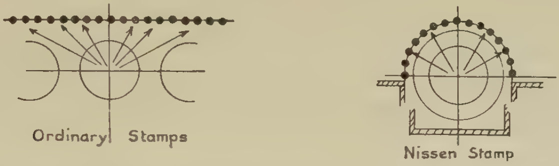


Fig. 108.

Nissen Stamp.—Diagram showing Splash against Screen. Compare the semicircular screen of the Nissen stamp and consequent radial arrival of the splash, with the straight screen and consequent oblique splash of the ordinary stamp (p. 158).

Crank-lifted Stamp

The principal limitation of all cam-lifted stamps is that they cannot be run above a certain speed. If lifted by crank a much higher speed is possible; the difficulty then is to arrange that the stamp falls freely and without hindrance from its connection with the crank. The Holman stamp accomplishes this by having an air-cushion between the driving mechanism and the stamp (Figs. 109, 110). With it, the power is supplied from an overhead shaft, through cranks and connecting-rods, to a vertically-guided cylinder which moves up and down at every revolution of the crank. Through this cylinder the stamp stem passes, carrying a piston which fits the cylinder. In the sides of this cylinder are two rows of holes, one nearer the top and the other nearer the bottom. As the cylinder moves upward from its lowest position, the piston remains without movement until the air trapped and cushioned below the lower air-holes is capable of supporting the stamp. It then flies up and the stamp with it, but before finishing this flight, finds the cylinder on its way down. The air above the upper holes then becomes trapped and compressed till the stamp begins to fall, when it expands to hasten that fall. To suit the conditions of a worn shoe and die another two rows of holes are bored lower down than the first set, these holes being plugged till they are required, when the plugging is transferred to the original set. In consequence of the delay in the lift and of the accelerated fall, there is a perceptible interval of rest upon the die, even though these stamps make as many as 140 drops per minute.

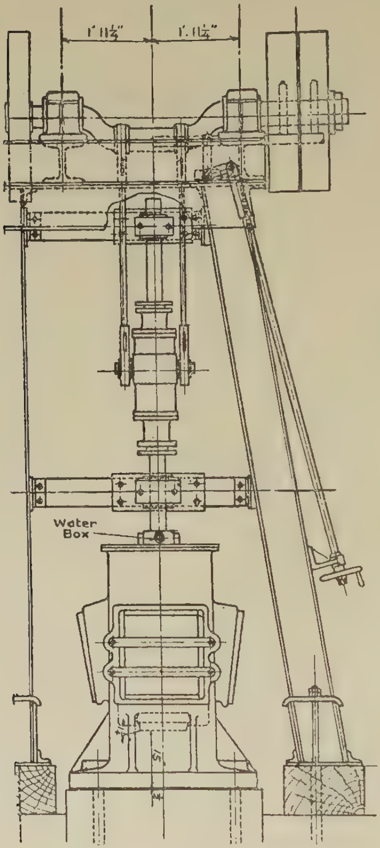


Fig. 109.

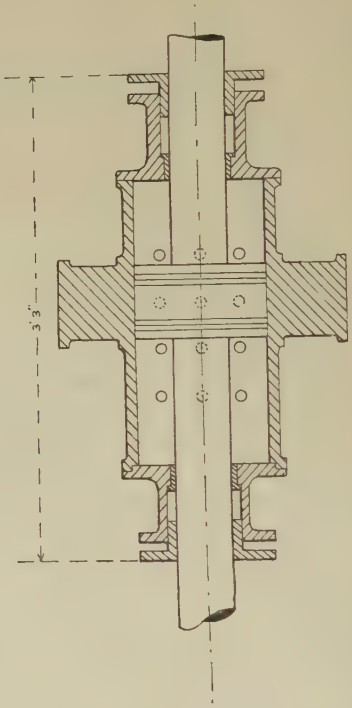


Fig. 110.

Crank-lifted Stamp (Holman Bros., Camborne, Cornwall) (Fig. 109).—The front elevation shows fast and loose pulleys, fly-wheel, two connecting-rods, air-cushion cylinder, upper and lower guides to piston rod which is the stamp stem, mortar box with front and side discharges, and wheel to operate the belt-shifting gear. The framework is of steel girders, the right half of a 2-stamp framework being illustrated (p. 159).

Air Cushion of Crank-lifted Stamp (Fig. 110).—Projecting from the cylinder wall are the two trunnions which are engaged by the two connecting-rods depending from the crank. In the cylinder wall are four rows of holes, worked in pairs—the first with the third and the second with the fourth—the holes of one pair being plugged while those of the other are operative (p. 159).

The shoe is 12 in. in diameter and the falling weight about 1200 lb. Each stamp works in its own mortar box, which is a casting having screens on three sides when rectangular in shape, or a semicircular screen when round in shape, the feed being at the back. This box stands on a heavy concrete base.

One of these stamps will crush 25 tons of hard ore from $2\frac{1}{2}$-in. ring to 20 mesh per day at a consumption of 15 h.p.; or roughly 150 lb. per h.p. hour at a size-reduction of 100.

The speed at which these stamps run is determined by the capability of the containing structure to withstand excessive vibration. They are usually run in pairs within a steel framework converging towards the top, where the crank shaft is borne. The number of drops is about 135 per minute, and the height of drop about 12 in. Though usually no special means for turning the stamp is provided the shoe wears quite evenly, the stamp apparently turning of itself.

In consequence of the high speed, these stamps require considerable repair, but against this may be set their greater capacity for the same dead weight of machinery. They are largely used in Cornwall upon tin ores, and have been used elsewhere.

The Steam Stamp

This stamp applies the principle of the steam hammer. It was evolved in the Lake Superior copper-district where it has received its greatest development; in fact it is little employed elsewhere. Ore at Lake Superior is mined in large blocks which at the surface are broken by large jaw-breakers to about 4 or 5 in., the material being then fed to the steam stamps. This ore is either hard conglomerate or softer amygdaloid. In either material the copper occurs native in lumps, stringers, grains, and flakes, which have to be freed from the gangue in which they are irregularly distributed. From the breaker product, some of this copper can be picked out as it passes along the chute to the stamp. For subsequent concentration the remaining ore was formerly stamped through $\frac{3}{16}$-in. round holes. Latterly, however, it has been found that $\frac{5}{16}$-in. holes with the conglomerate and $\frac{5}{8}$-in. holes with the amygdaloid, give a larger capacity to the stamp and an equally good recovery of the copper.

The stamps themselves consist essentially of a stem elastically coupled to a piston rod at its upper end and having a removable shoe below (Fig. 111). This stamp is lifted and then forced down upon a die in a circular mortar-box which, in the large sizes, is about 6 ft. diameter by 2 ft. 6 in. deep outside, and 4 ft. by 2 ft. inside. The box sits on a heavy anvil with a square base, this anvil in turn sitting upon a huge concrete block. The steam cylinder or cylinders are supported at the apex of a skeleton pyramid of columns which rise from the four corners of this block. Across these columns come the members which guide the stem.

The engine may be single-cylinder or compound, the single or the high-pressure cylinder being about 20-in. diameter and 24-in. stroke. The steam pressure is about 120 lb. and cut-off generally about half-stroke. The valves may be ordinary slide valves or Corliss valves. They are operated by an independent mover, and adjusted to provide a cushion of entrapped steam above the piston at the end of the upward stroke. For that stroke much less steam is given, or it may be brought about by a smaller cylinder in tandem below the main cylinder and using lower-pressure steam. Such an auxiliary cylinder would be always under steam even during the downward stroke.

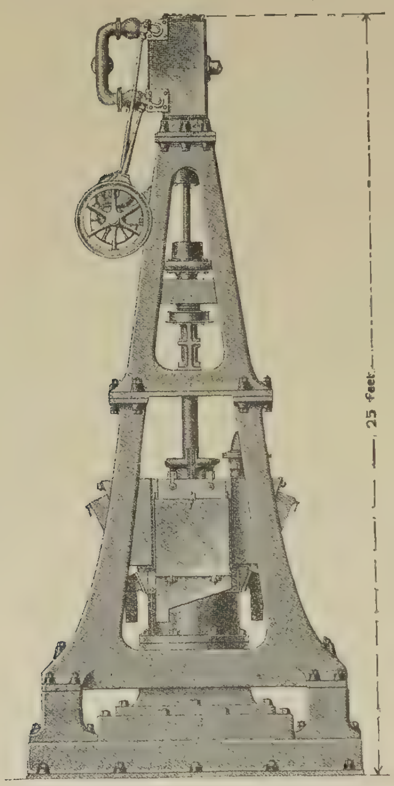


Fig. 111.

Steam-driven Stamp.—Side View. Cylinder at the apex of a pyramidal framework containing the mortar within its base. Steam admission controlled by independently operated valves. Piston rod ends in a dash-pot connection with the upper end of the stamp stem. Below this elastic connection is the pulley by which stamp is independently rotated. Front and side discharges provided with splash shields and discharge shoots. Mortar-box circular and upon a heavy iron anvil (p. 161).

The piston rod is connected to the stamp stem by a dash-pot joint, this providing the necessary elastic connection. The stem of 8 in. diameter is flattened at the bottom to receive an ovoid shoe attached to it by a dovetailed shank. Though the falling weight of the complete stamp is considerable of itself, the steam provides the essential crushing force. If the pressure of the steam be 120 lb., the diameter of the cylinder 20 in., and steam be cut-off at half stroke, that is, at 12 in., the work put into the stamp per stroke by the steam would be 37,700 foot-pounds. If the falling weight be 2000 lb., then as this drops 2 ft. the additional energy due to gravity would be 4000 foot-pounds, making 41,700 foot-pounds altogether. Neglecting the expansive force of the steam on the one side and the friction on the other, this figure represents approximately the energy available for crushing.

Using a screen of $\frac{5}{8}$-in. round holes and making 100 blows per minute such a stamp would have a capacity of about 400 tons per day; or with a $\frac{5}{16}$-in. screen about 240 tons. The diameter of the die is 22 in. and the height of discharge about 1 foot. Around the die, but separated from it by a space, comes a circular liner. This liner fits into the solid box upon which is reared the housing holding the screens.

Since the native copper is often in such pieces as cannot be discharged through the ordinary screen some additional discharge is also necessary. Formerly each stamp was stopped while such copper was taken out, but now it is usual to discharge it either down through pipes up which a sufficient current of water rises to keep back lighter material, or by means of small jigs, 4 × 6 in. in size and four in number, two on either side of the box. These jigs are disposed at about the same height as the ordinary discharge and have sieves with 1-in. holes; they are worked by plungers in the ordinary way and discharge from them is continuous. A substantial proportion of the copper recovered is separated in this way.

The feed to these stamps is regulated by hand. A man at the chute pulls down more ore as such is required and at the same time picks out copper, chips, etc. It is usual to run with 4—6 in. of material upon the die so that wear of the die is not great. When the stamp has worked its way down below a certain level a bell is struck from the stem. In order that the complete circular surface of the die may regularly be engaged by the ovoid shoe, the shoe is made to revolve about once every four blows by an independently-driven belt encircling a pulley on the stem. This relation of shoe and die permits ore to place itself upon the disengaged portions of the die to await crushing. Their ends having obviously considerably more work to do than the central portion, the shoes are often made with a depression at the centre. Shoes, dies, rings, and liners are usually made of chilled cast-iron, as this material has been found to be the most economical under the conditions of heavy wear at Lake Superior. About 5—7 tons of water are used per ton of ore; there is a good deal of splashing in the box, the shoe being lifted right out of the water after the blow. The capacity reckoned in terms of power consumed, is about 240 lb. per h.p. hour when $\frac{5}{8}$-in. screens are used. Power is responsible for about two-thirds of the total operating-cost, this being 8d.—9½d. per ton crushed. The lower costs are with the amygdaloid.

Beater or Hammer Mills

In beater mills not only is crushing effected by impact but the energy of that impact exists rather in high velocity than in great weight, and the material instead of lying upon a fixed die is broken largely in mid-air. Accordingly, these mills are operated dry. They are chiefly employed on non-metalliferous and softer materials.

Hammer Mill.—In this mill two dozen hammers or so, each weighing about 25 lb., are attached by chains at close intervals along a shaft making

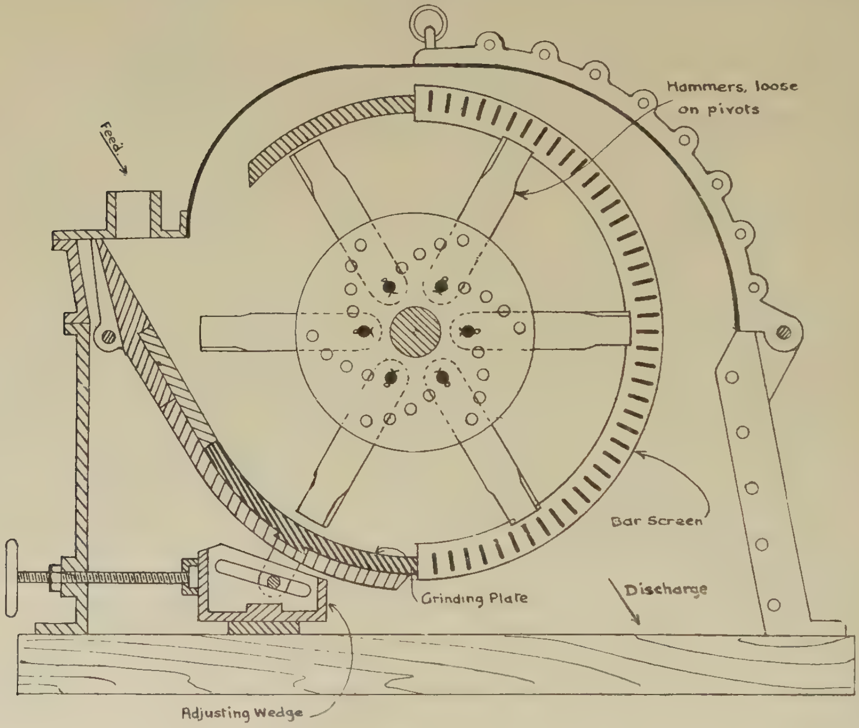


FIG. 112.

Beater or Hammer Mills.—**Beater Mill** (Diagrammatic Section). The illustration shows the feed to fall upon a grinding plate against which much of it is crushed. As this plate wears it can be renewed, while as the beaters become shorter they can be pivoted farther out from the central shaft. When the machine is still the beaters hang limply downwards (p. 165).

about 300 r.p.m. This shaft revolves inside a cage of grizzly bars spaced $\frac{3}{16}$—$\frac{3}{8}$ in. apart, making about 60 r.p.m. in the same direction as the shaft. The length of chain attaching the hammer is such that this latter swings just clear of the grizzly bars. The ore, which may be of ordinary lump size, is fed centrally at one end, and discharged through the grizzly wall. An ordinary mill of this type working upon suitable material is stated to have a capacity of about 15 tons per hour through a $\frac{3}{8}$-in. aperture at a power consumption of 30 h.p., figures which roughly represent a size-reduction of 10 and a capacity of 1000 lb. per h.p. hour. At one mine this machine was employed in the place of breaker and stamps to crush gold ore; and at another mine to strip the valuable matrix from the worthless pebbles of a hard-cemented gravel, the former discharging between the bars, the latter discharging as oversize at the end. A disadvantage is the amount of dust created.

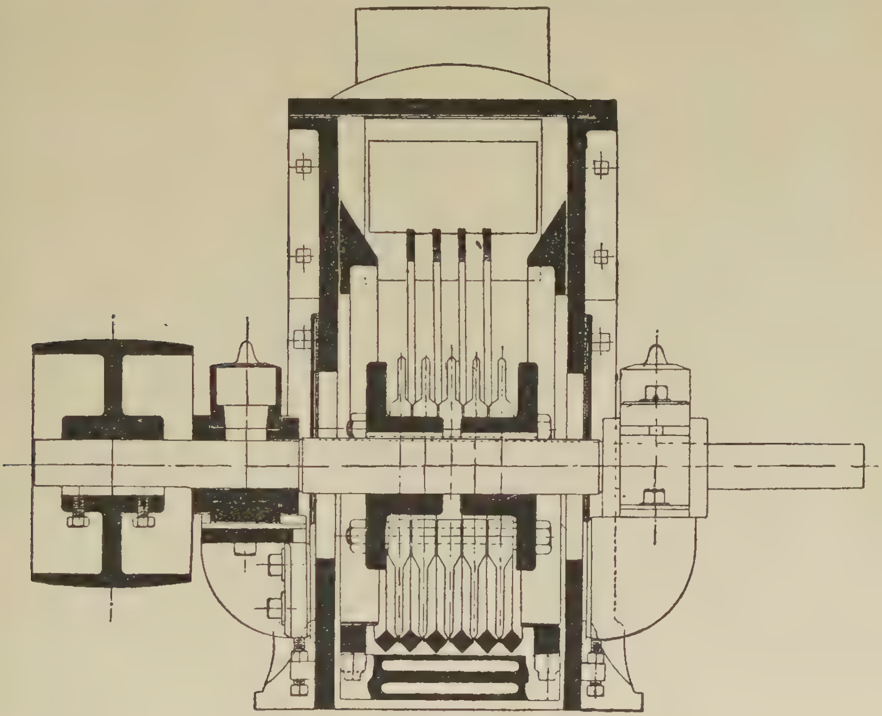


Fig. 113.

Beater Mill.—Sectional End Elevation of the Gannow Pulverizer. Five flails are shown extending downwards into the grid of diamond-shaped bars. Above the central shaft four other grid-bars are seen, so spaced that the beaters swing freely between them. These bars are on the feed side of the machine, where they are laid at an angle of about 45° to hold back the coarser pieces until the beaters, striking through the bars, have broken these pieces small enough to pass into the whirlwind circle within (p. 167).

Beater Mills.—In these mills short bars, attached by pins to naves upon a central shaft, and swinging freely on these pins, deliver a blow like that given by the threshing flail. As a rule there are three to six bars

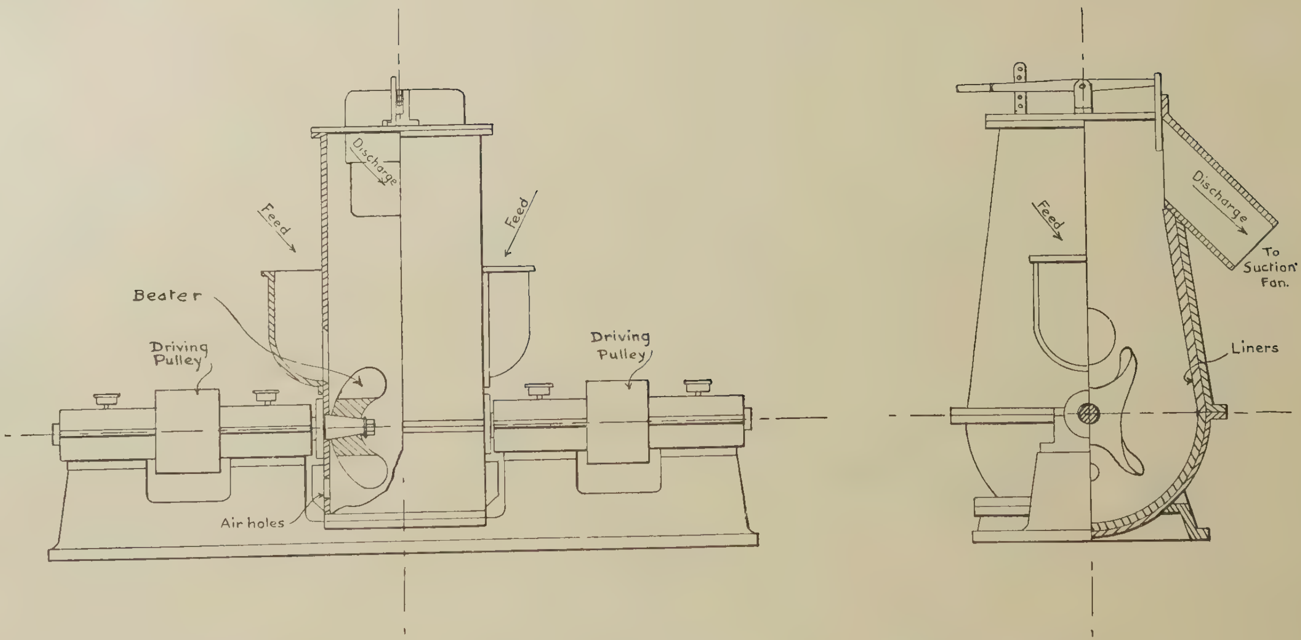


Fig. 114.

Blade Disintegrator.— Half-sectional Side and End Elevations of the Cyclone Mill (p. 167).

(Cirkel, *Chrysotile Asbestos*, Canadian Department of Mines, 1910, p. 134.)

spaced regularly around the circle, four to six of such circles closely following one another along the central shaft. The ring wherein these beaters meet the ore is formed by solid breaking-plates on the side where the feed enters, and by a semicircular barred screen on the opposite or discharge side. In turn, the whole mill is enveloped in a housing (Fig. 112).

The Quenner mill of this general type has been used to break cemented auriferous gravel where, for lack of water, dry washing has been necessary, while the Gannow pulverizer is used in this country to make the fine stone-dust strewn in colliery ways, the feed being generally prepared by a breaker (Fig. 113).

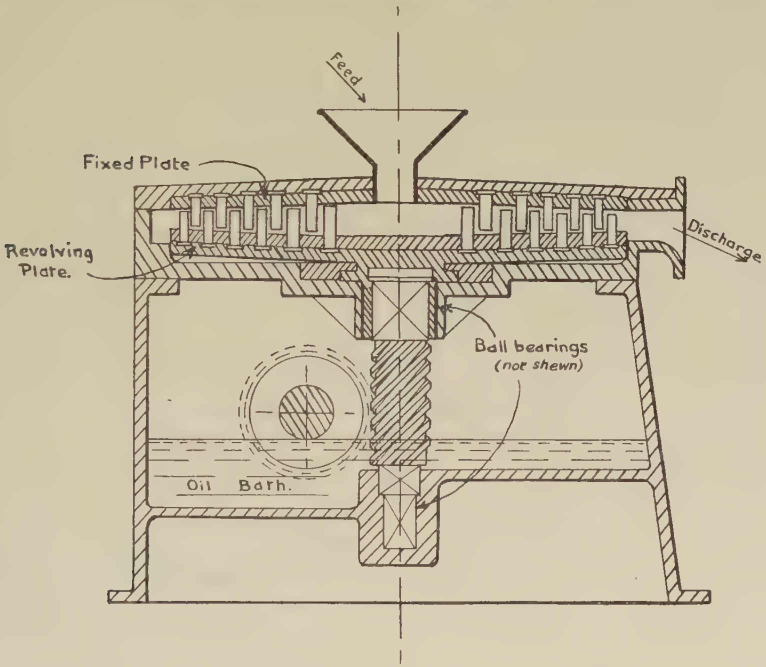


Fig. 115.

Pin Disintegrator.—Sectional Elevation of the Kek Grinding Mill (p. 168).

Blade Disintegrators.—The Cyclone mill, used chiefly in crushing asbestos rock in Canada, consists of two beaters of the screw-propeller type driven at a speed of 2000—2500 r.p.m. in opposite directions, in a cast-iron chamber into which material of walnut size is fed. The whirlwind created by the beaters hurls the particles against one another with such violence that they are reduced to a size such that they can be sucked away by a fan. The blades, made of chilled iron, wear out rapidly and have to be replaced every ten days or so. Of relatively hard rock 25—30 tons can be crushed in ten hours, this amount increasing to 40—50 tons when the rock is softer (Fig. 113).

Pin Disintegrators.—In these mills two discs with steel pins projecting in concentric rows, face one another so closely that the pins practically overlap, the concentric rows of one disc fitting in between those of the other (Fig. 114). Each disc is separately and oppositely driven, while through the centre of one the material is fed. Flying outward, the pieces are struck first by one row, and then from the opposite side by the next succeeding row, this treatment being repeated two or more times according to the reduction to be effected. Beyond the outside row, discharge takes place into a housing shaped to provide a suitable exit at the bottom. These machines, which may be horizontal or vertical, vary from 2 ft. to 6 ft. in diameter, and have pins from 4 to 12 in. in length; the driving shaft of the smaller machines makes about 600 r.p.m., and that of the larger 200 r.p.m., the disc revolutions being in the thousands. They take from 4 to 40 h.p. and crush 4 to 40 tons per hour of coal, phosphate, shale, chalk, etc. With harder material the wear on the pins would be excessive.

CRUSHING MACHINES COMPARED

The capacities given in the foregoing descriptions of crushing machines may be collected into the following statement:

Machine.	Size.		Size-Reduction.	Weight Crushed per h.p. hr.	Daily Capacity. Tons.	Cost per ton.
	Feed.	Discharge.				
Breaker . . .	..	2″	4	2000 lb.	120—1200	2d.
Disc breaker . .	2″	3/8″	5	1250 ,,	120—600	3d.
Rolls, coarse . .	2″	1/2″	4	1000 ,,	100—400	6d.
,, fine . . .	1/2″	10 mesh	10	600 ,,	100—400	12d.
Pendulum mill, wet .	3/4″	16 ,,	24	166 ,,	30—60	15d.
,, dry .	1½″	30 ,,	90	80 ,,	10—20	24d.
Edge-runner, slow .	2″	30 ,,	120	175 ,,	20—40	12d.
,, fast .	3/8″	20 ,,	15	180 ,,	50—80	12d.
Grinding pan . .	1/20″	50 ,,	5	275 ,,	20—25	18d.
Tube-mill, short . .	1/6″	60 ,,	20	200 ,,	50—250	12d.
,, long . .	1/30″	120 ,,	10	80 ,,	25—100	30d.
Ball-tubemill . .	2″	10 ,,	40	400 ,,	50—400	8d.
,, ,, . .	2″	40 ,,	160	150 ,,	40—200	12d.
Ball-mill . . .	3″	25 ,,	150	180 ,,	40—100	24d.
Stamp, gravity . .	2″	3 ,,	12	350 ,,	8—20	12d.
,, ,, . .	2″	15 ,,	60	200 ,,	5—8	20d.
,, ,, . .	2″	30 ,,	120	125 ,,	2—4	36d.
Stamp, steam . .	4″	1/2″	8	240 ,,	250—400	9d.

The figures in this statement indicate, as would be expected, that the finer the crushing stage the greater the consumption of power for the same mass crushed; compare, for instance, the work done by breakers, namely, about 2000 lb. per h.p. hour, and that done by tube-mills grinding to slime, about 80 lb. per h.p. hour. They indicate, again, that with machines operating at a large ratio of size-reduction, as for instance, stamps, ball-mills, and slow-speed edge-runners, the capacity does not suffer so much as might be expected; their capacity in fact appears to be determined largely by the fineness to which crushing is taken, and, accordingly, such machines show to best advantage when there is a considerable proportion of large material in the feed.

What these figures do not disclose, however, is the "mechanical nature" of the product. In this matter, sizing analyses show that those machines whose crushing is arrested, such for instance as breakers and rolls, yield much material approximating in size to the set, whereas pulverizing machines deliver a product most of which is far finer than the limiting aperture; further, that, generally speaking, those pulverizing machines which have a quick discharge, such as the short or conical tube-mill, the wet pendulum mill, and the fast edge-runner, give a more or less granular product, while those with a tardy discharge yield a greater proportion of exceedingly fine material.

Comparing the different machines in respect to their range of application, the stamp mill is the widest applied. Arranged with high drop it is suitable for the hardest and toughest ore, while with low drop it is equally suitable for soft ore. Working with low discharge and coarse screens it has a high capacity, delivering a product which contains a fair proportion of the finest material. Working with high discharge and fine screens, though its capacity is much less, it delivers a material of a very fine average size. These last were the conditions under which stamps early came into favour; the recovery of gold from auriferous quartz by amalgamation demanded a fineness unnecessary at that time with base-metal ores. In reducing such hard material to this fineness the stamp was particularly well fitted. Later, when the adoption of cyanidation demanded still greater fineness, the stamp, by reason of the amount of fine material it made even when crushing coarse, remained an efficient primary crusher, to prepare the material for regrinding.

In its constitution the stamp mill has a multitude of parts. In operation also many adjustments have to be made and much careful attention and strenuous work are required to maintain high working efficiency. It also makes a deafening noise and sets up much vibration. Yet anything wrong is quickly observed, and being confined to a relatively small unit may be set right while the mill as a whole continues uninterruptedly. The stamp mill accordingly has the convenience of a small unit while possessing the capacity of a large unit. Stamps are reliable, that is to say, are easily kept running 95 per cent of full time, and the attendant labour is readily trained; but they are costly to purchase and to install.

Of somewhat similar usefulness is the slow-speed Chilian mill which appeared contemporaneously with stamps, but elsewhere and for the recovery of silver rather than that of gold. Crushing by the dead weight of heavy rollers moving slowly around a circular track, these machines are heavy in relation to capacity, expensive in first cost, and occupy much space. On the other hand, they are smooth-running simple machines, giving rise to little noise or vibration and necessitating but simple foundation and driving mechanism. Consuming but little power when running light, and automatically adjusting themselves to wear, depth of feed, etc., their operating cost is substantially lower than that of stamps while their product has much the same general character. For very hard ore they are not, however, so suitable as stamps, nor are they convenient for large installations.

Except with tin and native copper, stamps have never had any great vogue in crushing base-metal ores, this having been the particular province of rolls. Rolls make a minimum of fine material and therefore permit a maximum of coarse concentration, to the advantage of total recovery. In regard to the character of the ore to which they are suited, rolls have not the same wide range as stamps; they are neither suited to very hard and tough ore, on the one hand, nor can they handle clayey ore, on the other. Nor have they the same range in respect to size of product. Though, like the stamp, capable of taking the breaker product, rolls are rarely set to effect in one operation a size-reduction greater than 8, while under some conditions 100 is normal for stamps. To effect any moderate size-reduction with rolls there must be stage crushing. Even then, difficulties of keeping the roller-face true are such that the efficient crushing of fine material is impossible, and rolls are not often set to crush finer than 10 mesh. They nevertheless have great capacity and are therefore particularly useful where large quantities require but little preparation for the next-following step; not, however, when it is required eventually to reduce all to slime, but when it is required to concentrate by water and the production of fine material must be avoided. In such circumstances crushing from 2 in. to $\frac{1}{4}$ in. by rolls would be cheaper and more advantageous than by stamps.

In first cost also the roll is much the cheaper machine; it is mechanically simple, requires little height or space, needs but an inexpensive foundation, and makes no such noise as characterizes the stamp mill.

Rolls differ again from stamps in that they are generally operated dry. Discharge with them needs no assistance from water and the extra wear of the crushing faces which accompanies wet crushing is thereby avoided. Stamps, it is true, have also been operated dry, but always at the expense of capacity. Working dry, rolls are particularly suited to the crushing of oxidized ores, some of which being extremely friable would readily form undesired suspensions in water.

With the need of improved recovery from base-metal ores came the necessity for finer grinding than economically possible by rolls. For this work wet-crushing pendulum mills were first favoured, these mills by reason of their quick discharge delivering a granular product in good condition for water concentration. To it also the Chilian mill was applied, being made suitable by increasing the pivotal action and by lowering the discharge, and made of greater capacity by increasing the number of rollers and the speed. With these alterations the Chilian mill was preferred to the pendulum mill where large capacity was desired.

In the meantime, the necessity for the finest grinding of precious ores having arisen, the pan used in amalgamation was modified to make grinding its essential purpose. Such grinding pans were simple and satisfactory, but being small and the wear of the grinding surfaces being relatively great, they gave way to tube-mills which, in addition to large capacity, were capable of more completely reducing the material fed to them. Any desired product being obtainable by working these mills in closed circuit with a classifier, they were used not only for the finest comminution but also to give finely-granular material. For this work shorter mills were found to be not only sufficient but more suitable, especially when equipped with scoop discharge ; the shorter the mill the more granular the product. In this direction also the conical tube-mill found application, since though discharge was still by overflow, it was facilitated by the design.

These modified tube-mills are thus granulators, and in consequence particularly suitable for the further comminution of base-metal ores, to which work they have latterly been widely applied, in many places actually replacing pendulum and fast-running Chilian mills. Where, however, water concentration follows, they cannot so satisfactorily be worked in circuit with classifiers, or the heavy mineral returning with the underflow would be ground impalpably ; under such conditions the portion to be reground should be separated by screens.

Though ball-mills discharging through fine screens had long been successfully applied to the dry-crushing of the breaker product, wet ball-mills to effect a similar size-reduction were late in being developed, chiefly because of excessive wear of the lining, balls, and screens, then available.

The success of tube-mills, however, turned attention again to the possibilities of similar mills for coarse material, and to-day ball-tubemills working wet have proved themselves most successful provided the material fed to them were of proper size.

There is little doubt that where large capacity is desired ball-tube-mills will be successful on base ores generally, and on precious ores to a large extent. In them the many desiderata of a good crushing machine are combined: they are mechanically simple and have a minimum of parts; they are small and occupy little space; they are relatively light in weight, and low in cost; they are reliable and have a long life; they are operated cheaply, and require relatively little water. The point which largely determines their use is the wear of the balls and lining. If the ore is too hard this wear will be excessive and their many good points may be outweighed.

Finally, it would appear that in the future the great vogue of stamps will diminish even with precious-metal ores. It may be that the position of rolls with base-metal ores is similarly threatened by the advent of the disc breaker, a machine which, though originally introduced as a fine preliminary breaker to reduce material of 4—5 in. to 1 in., is now crushing 1—$\frac{1}{2}$ in. material to pass a $\frac{3}{8}$-in. aperture, doing which work it has a high capacity.

Some machines, such as the breakers, the Griffin mill, and the disintegrators, only work dry. The same may perhaps be said of rolls, the small amount of water sometimes used hardly constituting wet crushing, which implies crushing in water and discharge by water. Other machines, such as ball-mills and tube-mills, work both wet and dry, the dry tube-mill, for instance, being an ideal dustless machine. Finally, some machines, such as stamps, Chilian mills, Huntington mills, and grinding pans, always work wet.

Dry crushing is necessary where dry-magnetic concentration, or where roasting or other heat treatment, follows; sometimes also its adoption is unavoidable owing to lack of water. Compared with wet crushing there is less wear of the crushing faces, but owing to dust often greater wear of bearings. Unless provision is made to collect it, this dust may not only be a nuisance but a direct loss of ore. When crushing dry to a fine size, capacity usually suffers; in addition, the ore must first be dried if it contains more than about 1$\frac{1}{2}$ per cent of moisture. On the other hand, the dry-crushed product is better for subsequent water concentration and for leaching, perhaps because less material of slime-size is produced. The dry crushing of friable and weathered ores also remains possible when wet crushing might be impossible by reason of the large amount which then would go into an undesired suspension.

In the finer stages wet crushing allows more easy discharge and consequently greater capacity, more easy sizing, and more easy conveyance of the product; in those stages it is consequently the rational and normal practice. In the coarser stages, on the other hand, dry crushing does not produce a great amount of dust, the material is large enough to be screened dry, while no assistance from water is required in discharge or could be rendered in conveyance; dry crushing accordingly is practised.

CHAPTER IV

COMMINUTION: THE CRUSHING SYSTEM

The sequence of machines by which complete comminution is accomplished may be described as the Crushing System. The extent of this system and the machines employed depend primarily upon the nature of the ore and the desired fineness of the crushed product.

General Considerations

Most ore-minerals are brittle when compared with their associated gangue. Some, such as galena, chalcopyrite, chalcocite, argentite, the gold and silver tellurides, wolframite, etc., and most oxidized minerals, are brittle to the extent that they go to pieces while much of the gangue remains relatively coarse. Others, such as cassiterite, pyrite, blende, etc., though not giving way to the same extent, generally display themselves more brittle than the bulk of the gangue.

Generally speaking, screen analyses of crushed ores containing the more brittle minerals will show the fraction finer than 200 mesh, to assay higher than the coarser fractions. Contrariwise, similar analyses conducted on ores containing the less brittle minerals would likely show this fraction to be lower in value than some of the fractions immediately coarser. In either case, however, if the fraction finer than 200 mesh were further divided by elutriation, it would probably be found that the very finest fraction, representing the clayey or aluminous portion of the ore, would be of distinctly low value.

Ores, however, generally consist to a large extent of gangue-mineral, represented by quartz, calcite, rhodonite, etc., and of country-rock, represented by sandstone, limestone, granite, rhyolite, andesite, schists, etc., and the resistance to crushing often comes rather from these than from the ore-mineral. Of such gangue-minerals quartz is hard and resistant, calcite soft yet resistant, rhodonite hard and tough. Among the sedimentary country-rocks, sandstone having low cohesion breaks relatively easily, limestone with more cohesion has nevertheless no great strength, while slate, protecting itself from complete comminution by its tendency to break in flakes, is somewhat stronger. Among the igneous rocks those of the acid group are hard to drill but easy to break, whereas basic rocks are soft to drill but tough to break, and the sequence of strength beginning with the strongest is something of the order: diabase, basalt, diorite, granite. Metamorphic rocks, such as quartzite, schist, etc., are generally very resistant, more particularly when, as so often happens in the neighbourhood of an ore-deposit, they are veined and impregnated with quartz. Rocks of a schistose or slaty structure are often tough.

Into this question of strength, texture also enters. A coarse-grained rock gives opportunity to break along crystal faces; a porphyritic rock is strengthened by the binding of the longer crystals; while a fine-grained rock resists by complete crystal interlock.

Strength is not entirely identical with hardness, or the acid rocks being harder than the basic would be the stronger, and the diamond being the hardest of all minerals would not be so easily powdered; strength lies in hardness and toughness combined. Though not identical, hardness is, nevertheless, such a factor in strength that the sequence in hardness is not greatly different from that in strength. Taking the metalliferous minerals previously mentioned, the sequence in hardness would be: cassiterite, wolframite, pyrite, blende, chalcopyrite, chalcocite, argentite, gold tellurides, galena. Cassiterite, the least brittle among them, is thus the most hard; and galena, the most brittle, the least hard; while quartz, harder than them all, is also stronger than all. The native metals, on the other hand, though soft, are tough and not easily comminuted. Hardness affects primarily the wear of the crushing surfaces and, secondarily, the power consumed in crushing.

In particular instances different ores have proved themselves hard or soft, as follows: the disseminated copper ores in the United States, consisting of altered monzonite and schists, are relatively soft; these, however, are shallow ores and not entirely fresh. The banket ore of the Witwatersrand, South Africa, consisting of pebbles fast cemented in a quartzitic matrix, is hard and tough. The quartz-tourmaline impregnated tin ores in Cornwall are hard and very tough. The flinty cupriferous conglomerate of Lake Superior is hard and tough, while the amygdaloid in the same field is relatively soft and easily crushed. Finally, the plumbiferous quartzites of the Cœur d'Alene district, Idaho, may be classed as intermediate in strength.

As to the influence of the condition of a rock upon its resistance to crushing, if a rock be fresh it is stronger than in any weathered condition; ore from depth is therefore harder and tougher than shallow ore. Wetness or dryness apparently have little effect upon the strength of the usual run of ores, though much wetness may seriously inconvenience the dry breaking with which comminution usually begins. Frozen ore, however, is more easily broken, which is readily understood, seeing that freezing has a bursting effect upon anything containing water and practically all rock holds some water in its pores. Heating and quenching are well known to have a weakening effect upon the cohesion of ore, and to this end have sometimes been applied. The winning of the ore by blasting also weakens the cohesion, so that the ore mined has not the same strength as when *in situ* underground. It is, however, much cheaper to break ore by machines on surface than to shatter it completely by blasting underground.

Determinations of the crushing strengths of pieces of fresh rock and ore have shown sandstone to have in round figures a strength of about 7500 lb. per square in., limestone 10,000 lb., granite 20,000 lb., soft ore below 25,000 lb., and hard ore above 25,000 lb. per square inch. Such determinations have usually been made upon cubical pieces lying with smooth face upon a smooth plate against which the pressure was applied at right angles, that is vertically downwards. Such good conditions for resisting fracture are, however, not realized in comminution, the ore pieces, both in their departure from the cubical shape and in their imperfect seating, being more easily broken ; tensile as well as compressive stresses are set up, against which ore is relatively weak.

With respect to the deformation of ores under stress, ordinary rocks and ores are elastic up to a limit. Within that limit they suffer deformation from which they can return when the pressure is removed ; beyond it they fracture. No intervening stretch of permanent deformation without fracture has been recorded, the applied pressure rising continually till relief comes by fracture. Should fracture not take place the piece returns to its original shape, being capable theoretically in so doing of giving back the work previously done upon it. It is this return of energy which causes the stamp to rebound. If the stamp could be caught at rebound and further lifted this energy would be utilized, but the danger of camming does not permit such close running. In jaw breakers, though such energy conceivably assists in the retirement of the jaw, that assistance is inappreciable. Accordingly, unless fracture is secured such elastic deformation represents lost work. It is largely avoided when the blow is so smart that the piece has not time to extend before being overtaken by fracture.

Where water concentration follows, the maximum release of the mineral is desired while excessive comminution must be avoided. Where flotation concentration follows, the maximum release of the mineral is desired but no similar necessity exists to avoid fine comminution. Finally, where amalgamation and hydro-metallurgical operations follow, the maximum exposure of the mineral is desired, making the finest possible comminution welcome and often necessary.

The extent to which crushing is carried also largely depends upon the ore value. Crushing is an expensive operation and generally the largest single item of cost in ore-dressing. Moreover, the finer it is carried the very much more expensive it becomes. With each case, therefore, a point is reached where the cost of further comminution is greater than the additional recovery resulting, this point marking the economic limit of crushing. This limit is reached earlier with low-grade ores than with rich ores, since the operating cost of a machine is independent of the valuable content. To keep the whole cost of a crushing system within the means of this content, the extent of crushing must vary with the value of the ore.

General Practice in Comminution

Dealing, firstly, with crushing in preparation for water concentration, stage reduction is the ideal, since, in addition to relieving the successive crushing machines of material already fine enough, it permits stage concentration, whereby not only may coarse mineral be withdrawn from further wasteful comminution, but opportunity may also be taken to discard worthless material. If both these possibilities be realized after each crushing, the material remaining to undergo the next stage is smaller in amount and the cost of crushing is to that extent reduced.

These full advantages of stage crushing are, however, only obtained when some at least of the mineral is coarse and stage concentration can begin early. It is applied notably at Anaconda, Montana, with copper sulphides; at Bunker Hill, Idaho, with galena; in Bolivia with cassiterite; and at many other places. Where the mineral grain is fine and no coarse concentration can be practised, the advantages are limited to the avoidance of excessive and unnecessary comminution, advantages which may or may not be sufficient recompense for the greater complication of stage crushing; in the coarser stages of crushing they would only be realized if the mineral were very brittle. As an additional point, it is generally held that in any crushing machine fine material protects the material which needs crushing, while receiving injury itself; the withdrawal of such fine material would therefore be advantageous.

Stage crushing necessitates, however, greater complication and greater variety of plant, drawbacks which make its full acceptance often of doubtful advantage. At Broken Hill, N.S.W., for instance, treating a complex lead-zinc-silver ore, crushing by rolls from $1\frac{1}{2}$ in. to $\frac{1}{8}$ in. is performed in a single stage, multi-stage crushing having proved uneconomical.

However varied the practice may be before concentration begins, there are very few plants where advantage is not taken of stage crushing afterwards, in the regrinding of middle products.

Dealing, secondly, with crushing preparatory to direct flotation, the ore suitable to this process is one wherein the mineral grains are small, and where, accordingly, stage crushing would not offer the opportunity to effect coarse concentration. Reduced to such a fine condition that the maximum size of particle is not too heavy to be floated, excessively fine material in ordinary amount does not interfere with the success of the operation. In these circumstances all that stage crushing has to offer is the opportunity to screen out the material already fine enough, and thus to send less material to the succeeding stage. To this saving may also be added the advantage that where, for successful recovery, the material must be crushed very fine, say to pass 80--100 mesh, the machines chosen for each step may be particularly efficient for that step, to the greater efficiency of the system. But where a less-pronounced degree of fineness is sufficient, say to 40—50 mesh, and a machine such as the ball-tubemill can be found to do the work in one step, the advantage of simplicity would more than outweigh any advantage which stage reduction could offer. Single-stage crushing for flotation is notably the practice at the Inspiration, Arizona, where the ore is reduced in one step from 1½ in. to 40 mesh.

Finally, where crushing is preparatory to amalgamation and cyanidation, the higher the proportion of very fine material the better the recovery of the precious metal, and the only advantage which stage crushing offers is, that by separating material already fine enough less material is sent to the succeeding stage, an advantage reinforced by the opportunity of employing machines particularly efficient for each stage. In this case, the separation of the coarse material for further crushing is made by water-sizing, a more convenient and cheaper operation than wet screening, but not so justifiable where concentration follows, because the mineral particles being heavier than the gangue would be continually returned to be reground.

The extent to which comminution is carried preparatory to amalgamation and cyanidation depends upon the character and value of the ore. With those gold-silver ores which have their valuable content most finely distributed and are of good grade, all the ore would be reduced to slime, a procedure impossible without stage crushing. Leaving breaking out of consideration, such crushing is usually done in two stages, stamps or ball-tubemills being followed by tube-mills. Exceptionally, however, three stages are employed, as at the Goldfield Consolidated, Nevada, where stamps, edge-runners, and tube-mills follow in succession.

With ore of medium grade crushing would have to be somewhat curtailed, and if stamps or ball-tubemills were used, they alone might suffice. Generally, however, stage crushing would be adopted, but the grinding machines would be set to achieve capacity rather than perfect fineness. In this class would also come those gold ores which yield much of their content to amalgamation, to the impoverishment of the material to be treated by cyanidation. Stamps followed by tube-mills producing sand and slime constitute the system which has become standardized in South Africa and Mysore, at the Homestake, Dakota, and elsewhere.

With poor gold-silver ores no great expense could possibly be incurred in crushing, and such ores become worked only when the coarsest crushing suffices. Amalgamation is then out of the question, while cyanidation can only be applied if the ore is porous. Such porous ores are exceptional and surficial, their porous nature being the result of weathering.

Coarse crushing in preparation for hydro-metallurgical treatment also suffices with certain low-grade copper deposits, notably at Chuquicamata, Chili, and at Rio Tinto, Spain, the two largest deposits of copper in the world.

Where concentration and cyanidation are practised upon the same ore, the crushing systems vary with the relative importance of these two treatments. Generally, the bulk of the material after removal of the concentrate is treated by cyanidation, and the ore is crushed as though for cyanidation. At Cobalt, Ontario, however, where the great bulk of the silver is obtained by concentration, and little is left for cyanidation to accomplish, coarse stage crushing for concentration is practised. On the Alaskan goldfield also, crushing is in preparation for the concentration of the valuable material, this material being subsequently reground for cyanidation.

Whatever the product desired, the complete crushing system may conveniently be divided into the following three broad steps :

Breaking, as applied to lump ore ;
Crushing, as applied to broken ore ;
Grinding, as applied to crushed products,

and suitable crushing systems for the different treatments would be as follows :

Water Concentration

	Ordinary Ore.	*Hard Ore.*
Breaking . .	Breakers	Breakers.
Crushing . .	Rolls or disc-breakers	Rolls or disc-breakers, stamps.
Grinding . .	Short or conical tube-mills	Short or conical tube-mills.

Flotation Concentration

Breaking . .	Breakers	Breakers.
Crushing . .	Ball-tubemills	Disc-breakers.
Grinding . .		Short or conical tube-mills.

CYANIDATION

	Ordinary Ore.	*Hard Ore.*
Breaking . .	Breakers	Breakers.
Crushing . .	Ball-tubemills	Stamps.
Grinding . .	Tube-mills	Tube-mills.

No one system need necessarily be the only system suitable to a particular ore. At Treadwell in Alaska, for instance, crushing for concentration of auriferous pyrite was accomplished at the Alaska Treadwell mine itself by stamps, at a cost of about 6d. a ton; at the Gastineau mine by rolls followed by short tube-mills, with equally satisfactory results; and at the Juneau, more recently but less satisfactorily, by ball-tubemills unaided.

In the selection of a system, consideration must be given to the particular experience of the available operators, to the capacity of the locality to supply spares, and sometimes also to local preferences.

The preliminary operation of breaking — and sometimes also the coarser stages of roll crushing—is often done in a building separate from that in which primary comminution is effected. Breaking can then be done at the mine, broken ore being transported to the mill, this procedure having the advantage that in ordinary mine-trucks such ore is much more conveniently transported than lump ore. The breakers can then be placed low down upon separate and solid foundations, and the mill building is spared the dust and vibration associated with breakers. While, finally, breaking may then be done during the day-time only, the power thus set free at night being a convenient provision for lighting the surface and mill. Where, however, operations are on a smaller scale, the breaker is often conveniently run from the same prime mover as the other machinery, with which consequently it would be housed.

The capacity of a crushing system is usually expressed in terms of daily tonnage, that is, in tons per 24 hours. That provided should depend upon the size of the ore-deposit. Up to a maximum of about 1000 tons per day there is advantage in great capacity. Should an ore-deposit warrant the daily treatment of a considerably greater tonnage than such a figure, it would be better to have two or more mills than greatly to increase the size of the unit mill.

The capacity expressed in pounds crushed per h.p. hour depends upon the fineness to which crushing is carried. Where all is reduced to slime this capacity will naturally be lower than when coarse crushing suffices. Taking first the extreme case of the reduction of a hard ore to slime, the following figures might be representative:

Breaker . . .	70 per cent of the ore.	2000 lb. per h.p. hour.
Stamps . . .	100 ,, ,,	200 ,, ,,
Tube-mills . .	75 ,, ,,	85 ,, ,,

Whence

2000 lb. would require by breakers . . .	1·0 h.p. hour.
2857 lb. (2000 × 100/70) would require by stamps	14·3 ,,
2143 lb. (2857 × 75/100) would require by tube-mills	25·2 ,,
2857 lb. of original ore	40·5 h.p. hour;

equivalent to about 70 lb. per h.p. hour.

Actual figures for the four-stage comminution at the Goldfield Consolidated,[1] Nevada, were:

	Daily Capacity.	Power.	Capacity per h.p. hour.
Breakers to 1½ in. . . .	600 tons	50 h.p.	1000 lb.
100 stamps to 4 mesh . . .	850 ,,	240 ,,	295 ,,
6 Chilians to 16 mesh . .	425 ,,	200 ,,	152 ,,
5 tube-mills to slime . . .	476 ,,	400 ,,	100 ,,
Total . . .	850 tons	890 h.p.	80 lb. per h.p. hour.

The other extreme would be where coarse crushing through rolls sufficed, of which the following figures might be representative:

Breaker . . .	70 per cent of the ore.	2000 lb. per h.p. hour.
Rolls	75 ,, ,,	600 lb. ,, ,,

Whence

2000 lb. would require by breaker	1·0 h.p.
2143 lb. ,, ,, rolls	3·57 h.p.
2857 lb. would require altogether	4·57 h.p.

equivalent to about 625 lb. per h.p. hour.

From the above figures it will be seen that to reduce all to slime would require somewhat more than 1 h.p. per ton of ore crushed per day, whereas when coarse crushing sufficed 1 h.p. would be sufficient for every 5 tons of daily capacity. Between, and even beyond these extremes, all the gradations demanded by varying conditions find record.

Naturally, the amount of power consumed greatly influences the cost of crushing. Renewals and repairs are, however, no less important, making together an item probably more decisive than power. Much depends upon the nature of the ore; to crush a hard, tough ore might well cost three times as much as a soft ore. The capacities of the separate machines and the size of the plant also greatly influence the total cost. Such total costs range from about 8d. per ton at the Alaskan mines in preparation for simple concentration; from about 12d. per ton with the disseminated copper deposits of Arizona for flotation; from about 20d. per ton on the Witwatersrand for the production of sand and slime for cyanidation; and from about 22d. per ton at the Goldfield Consolidated for reduction to slime, all these being large mines; to about three times as much on small mines.

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[1] Hutchinson, *M. & S.P.*, May 6, 1911, p. 616.

Particulars of the cost at the Goldfield Consolidated, the crushing system of which has already been detailed, were:

	Per Ton of Feed.	Per Ton of Original Ore.
Breakers	$2\frac{1}{2}$d.	2d.
Stamps	6·7d.	6·7d.
Chilians	10d.	5d.
Tube-mills	14·7d.	8·3d.
	Total cost per ton slimed	22·0d.

Work done in Crushing

When a piece of ore is broken in two, two additional surfaces are produced, one on each of the resultant pieces, and the additional surface exposed is consequently twice that of fracture. The work done in crushing was considered by Rittinger to be proportional to the area of fracture, and consequently to the additional surface exposed. This additional surface can be expressed in terms of the dimensions of the original piece.

Particle volume.	Number particles.	Particle dimension.	Total surface.	Surface increase or work done.
		The Cube.		
D^3	1	D	$6D^2$	
$\frac{D^3}{n^3}$	n^3	$\frac{D}{n}$	$6D^2n$	$6D^2(n-1)$
		The Sphere.		
$\frac{\pi D^3}{6}$	1	D	πD^2	
$\frac{\pi D^3}{6n^3}$	n^3	$\frac{D}{n}$	$\pi D^2 n$	$\pi D^2(n-1)$

Fig. 116.

Work done in Crushing.—Tabular statement using as index the additional surface produced. Summation of the work done in crushing a mass, assuming that work to be proportional to the additional surface produced (pp. 182, 183).

The size-reduction $D / \frac{D}{n}$ in either case $= n$.

Taking first the cube. A cube of side D would have a surface $6D^2$; if that cube were divided into a number of cubes of side D/n, n being the number of parts into which the side of the original cube was divided, the number of such cubes would be n^3, and the total surface $6(D/n)^2 \times n^3 = 6D^2n$, or an increase of $6D^2n - 6D^2 = 6D^2(n-1)$. (Fig. 116.)

Now, centring the considerations around a sphere of diameter D, the surface of such a sphere would be πD^2, and the volume $\pi D^3/6$. If that volume were divided into spheres of diameter D/n, n representing as before the size-reduction, the number of such spheres would be n^3 and the total surface $\pi(D/n)^2 \times n^3 = \pi D^2 n$, the increase of surface being $\pi D^2 n - \pi D^2 = \pi D^2(n-1)$. (Fig. 116.)

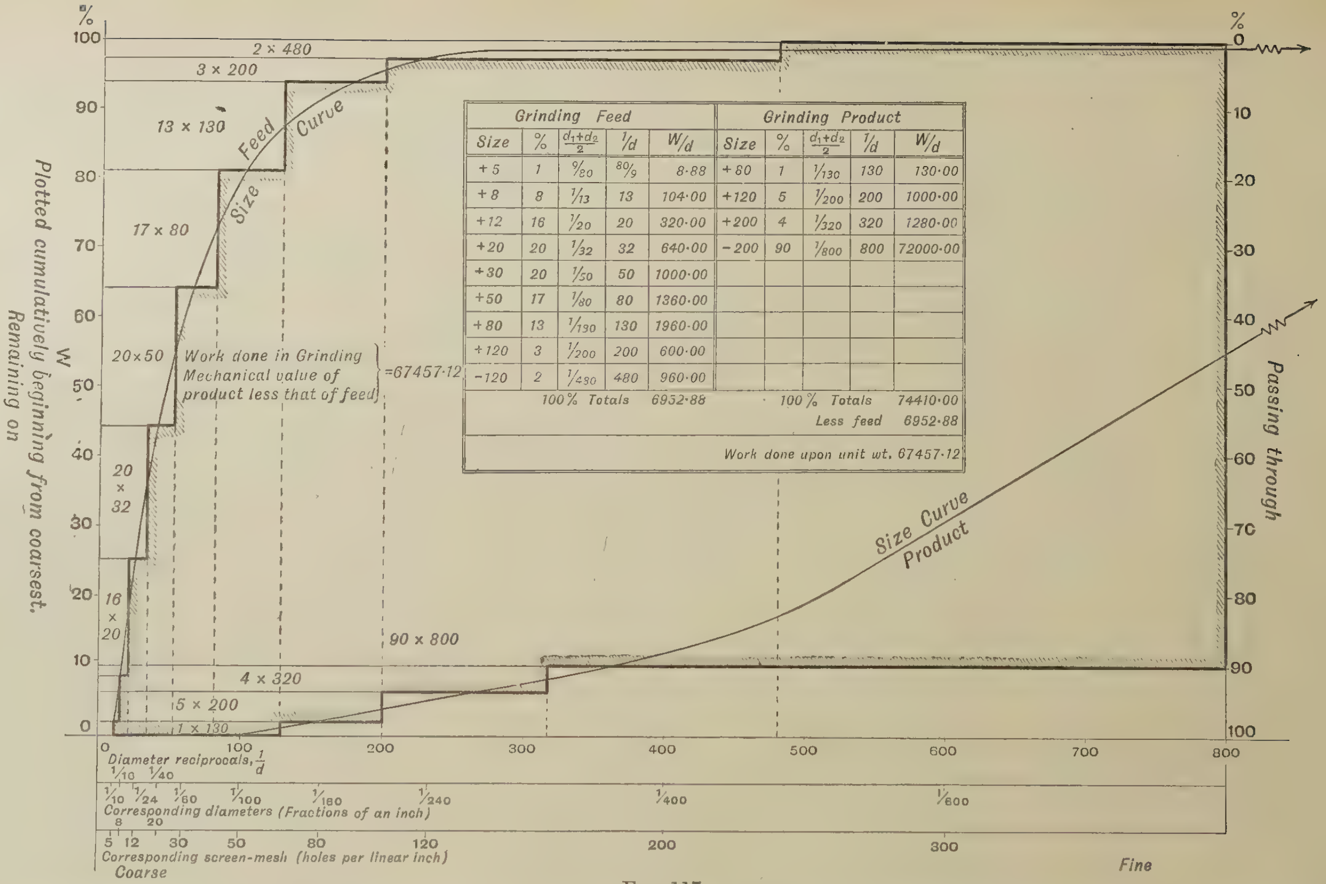
Actually, in crushing neither cubes nor spheres result, but irregular pieces, to cover which the general formula expressive of surface increase would be $C_1 D^2(n-1)$, D being the average dimension of the original mass. But n is equal to D/d—it is the ratio of size-reduction—introducing which the formula becomes $C_1 D^2(D/d-1)$ or $C_1 D^3(1/d-1/D)$. Of the factors in this formula D^3 is the volume of the mass under consideration, to which volume the weight is proportional. Accordingly, in the place of $C_1 D^3$, CW can be written, and the formula becomes $CW(1/d-1/D)$, or where D was infinitely large, CW/d, this again, where a proportional figure only were required, becoming W/d. This factor W/d may accordingly be taken to represent the work done in reducing a mass W from forming part of an infinitely large mass, to a condition with d as the dimension of the resultant particles. Ordinarily, most of these particles being roughly spherical, d may be taken to be average diameter.

This result may perhaps be more directly reached by the following considerations. The work represented by a mass of equal-sized particles is proportional to the total surface of such particles; the diameter of the particle being d, the surface of each particle is proportional to d^2; the volume of the particle being proportional to d^3, the number of such particles is proportional to $1/d^3$; and the total surface being proportional to the product of particle surface and particle number, is proportional to d^2/d^3, that is to $1/d$. Where the mass varied the work represented would vary proportionally, and in any mass of particles of diameter d the work done in bringing that mass to that condition of size from its presumed original condition of forming part of an infinitely large single mass would be represented by W/d. This expression, let it be repeated, accepts increase of surface as the index to the work done.

The work which any mass of particles represents may be described as the "mechanical value" of the mass. Under the form W/d this mechanical value has reference to reduction from an infinitely great size.

Keeping separate the two factors of which the above expression for work is composed, and plotting W as an ordinate, $1/d$ as an abscissa, this work is graphically given by the resulting rectangle.

Similarly, the work represented by a mass of mixed sizes would be given by the sum of a number of such expressions or rectangles (Fig. 117). To obtain the respective factors the mixed mass is divided by screens of



Grinding Feed					Grinding Product				
Size	%	$\frac{d_1+d_2}{2}$	$1/d$	W/d	Size	%	$\frac{d_1+d_2}{2}$	$1/d$	W/d
+5	1	9/80	80/9	8·88	+80	1	1/130	130	130·00
+8	8	1/13	13	104·00	+120	5	1/200	200	1000·00
+12	16	1/20	20	320·00	+200	4	1/320	320	1280·00
+20	20	1/32	32	640·00	−200	90	1/800	800	72000·00
+30	20	1/50	50	1000·00					
+50	17	1/80	80	1360·00					
+80	13	1/130	130	1960·00					
+120	3	1/200	200	600·00					
−120	2	1/430	480	960·00					
	100% Totals			6952·88		100% Totals			74410·00
							Less feed		6952·88
						Work done upon unit wt. 67457·12			

FIG. 117.

Representative Graph of Work done in grinding when measured by the extra Surface produced.—Plot of the calculations given on page 193. It will be found that the curve obtained by smoothing out the stepped line resulting from plotting rectangles, is the same as would be obtained by plotting direct the cumulative percentages respectively remaining on the screens. The stepped line is smoothed by drawing a curve to cut each vertical step at its middle point (pp. 183, 185, 186, 214, Fig. 119).

successively decreasing aperture into a number of sizes, the weights of which are expressed as percentages or decimal parts of the total weight. Such percentages or decimals represent W. To obtain the respective diameter-reciprocals the average diameter of the particles remaining between each two successive screens must be computed. The screens used for this purpose, described under 'Sizing,' are of woven wire, giving a square aperture. The actual maximum size of particle passing such an aperture is generally about two-thirds the side, but as this ratio is the same for all the sizes and at best only a relative result is in view, the side of the aperture may be accepted in place of the particle diameter. The average diameter of the particles between two successive sieves would then be the most simply obtained by taking the arithmetic mean of the two limiting apertures, that is $(d_1+d_2)/2$. That, however, would only be true if, in crushing, the numbers of particles of successive diameter were equal, whereas it is more probable that in a crushed product the weights of particles of successive size are equal, that is to say, the weight of fine material produced would be the same as the weight of the coarser or of any intermediate material. The latter assumption would make the average diameter the logarithmic mean of the two apertures, that is $(d_1-d_2)/\mathrm{Log}_e d_1/d_2$. This, however, is a refinement which need not be practised when the screens are as close together as they generally are in sizing analyses.

In graphic representation it is not even necessary to calculate the average diameters between screens, the actual screen apertures may be used instead; reciprocals of these apertures being laid off as abscissae, from their extremities ordinates are raised representing the cumulative percentages remaining on the respective screens (p. 212). The ends of such ordinates are upon a curve identical with the average curve of the stepped line resulting from plotting rectangles (Fig. 117).

More simply still, "screen mesh" being the number of holes per linear unit, may be taken to be half the aperture-reciprocal, and, where only relative figures are required, may be used in place of that reciprocal.

These reciprocals of aperture or diameter, however, increase enormously with increasing fineness. At the coarse end the reciprocal of an infinite diameter is zero, the reciprocal of the unit employed is 1, that of half the unit is 2, and so on, these figures also expressing the relative surface; at the fine end, on the other hand, reciprocal and surface are represented by such figures as 300, 400, and 500. With surface as index, the work represented by the finest portion of a crushed mass would accordingly be accounted as far exceeding that represented by the remaining and many times greater bulk of the coarser material. This seems unreasonable, casting doubt upon the index. Moreover, accuracy of result would depend upon accurate fractionation of the finest and impalpable portion, this accurate fractionation being impossible.

The large amount of work credited to the finest portion by this index is very clearly disclosed in the graphic representation, where at the coarse end the reciprocals are bunched together, while at the other end they rapidly extend in geometrically increasing steps towards infinity (Fig. 117). So rapid indeed is this extension that the diagram cannot be kept within reasonable limits without unduly cramping the coarse end. This, it is true, may be relieved by plotting logarithms of the reciprocals, but by doing so all claim on this basis to consider the resultant areas as representative of the work done must be abandoned.

When, however, instead of increase of surface, the number of unit volume-reductions is taken as the index of work done, which is the other view, such a logarithmic plotting produces an area which correctly represents that work. It is agreed that the force necessary to produce a fracture is proportional to the area of that fracture, that is, proportional to the additional surface exposed. But work is the product of force and distance. It is assumed by Rittinger that the distance moved through is the same for all sizes of particle, since fracture results by the rupture of molecular cohesion and this bond is independent of particle size. Cohesion, however, is not so simple of conception, and it is difficult to say at what point it begins or where it ends. Moreover, before fracture occurs a particle suffers deformation in proportion to its size, this deformation representing the distance through which the applied force moves. By those who hold Rittinger's theory, this fact is brought into harmony with that theory by assuming that though such variable deformation is true within the elastic limit, yet when fracture occurs it takes place along its own lines and is brought about not only by the direct application of force from outside but also by return of energy from within, as the mass resumes its original shape, the result being as though the distance moved through by the full force represented by the area of fracture was only that sufficient to break the molecular bond.

Stadler,[1] basing his argument upon Kick's law that "the energy required to produce analogous changes of configuration of geometrically similar bodies of equal technological state varies as the volumes or weights of these bodies," disputes the view that the surface exposed is the measure of the work done. He holds that though the area of fracture multiplied by a constant expressive of the strength of the material represents the force employed, this force must act through a certain distance of deformation to produce fracture, such distance being directly proportional to the diameter. Force therefore being proportional to surface, and distance proportional to diameter, the work done to break a single piece must be proportional to the volume of that piece, and not to the surface exposed. This view receives strong support from the fact that if bodies of the same material and shape but of different mass, be dropped separately from a height just sufficient to break them, it will be found that the height is much the same whatever the mass. Dropping from the same height the velocity factor in the energy developed remains constant, and that energy must accordingly vary as the mass; and since in each case the result is fracture, it may be assumed that the energy required to produce fracture varies as the mass or volume of the piece.

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[1] *Trans. I.M.M.*, 1910, Vol. XIX. p. 471.

Under this assumption, starting with a mass of dimension D, the work required to fracture this, say, into halves would be proportional to D^3; to fracture these two halves each of volume $D^3/2$ into quarters would require energy proportional to $2(D^3/2)$, or again D^3; to fracture these four quarters each of volume $D^3/4$ into eighths would require $4(D^3/4)$, or again D^3 (Fig. 118). The total work to reduce a mass of dimension D to eight masses of dimension D/2 would accordingly be $D^3+D^3+D^3=3D^3$. Proceeding further, the work required to fracture the eighths into sixteenths would again be D^3; the sixteenths to thirty-seconds again D^3; and the thirty-seconds to sixty-fourths again D^3. The total work necessary to reduce a mass of dimension D to sixty-four masses of dimension D/4 would accordingly be $3D^3+D^3+D^3+D^3=6D^3$. Similarly, the work necessary to reduce a mass of dimension D to five hundred and twelve masses of dimension D/8 would be $6D^3+D^3+D^3+D^3=9D^3$. The work accordingly increases by a unit amount each time the volume of the piece is halved: to reduce the mass to eighths each of dimension D/2, three units of work, namely to halves, to quarters, and to eighths, were required; to 64ths each of D/4, six units of work; to 512ths each of D/8 side, nine units of work; and so on. In other words the work increases in arithmetical progression while the number of pieces and, indirectly, the size-reduction increase in geometrical progression. But that is the relation between logarithms and antilogarithms. Accordingly the work done is proportional to the logarithm of particle number or to that of the ratio of size-reduction D/d. Of these two factors the latter is more directly obtainable, and in terms of the initial and final diameters it may accordingly be said that the work done is proportional to Log D/d.

Where the mass varies the work done would vary proportionally, and the work done in crushing a mass W from an initial diameter D to a final diameter d would be W Log D/d, a product represented graphically by the rectangle formed by plotting W as an ordinate and Log D/d as an abscissa. This mechanical value has reference to a finite original diameter.

Particle volume.	Number of volume-reductions.	Number of particles.	d Dimension of particle.	$\frac{D}{d}$ Size-reduction (ratio).	Log $\frac{D}{d}$	Units of work done.
D^3	0	1	D	1	0	0
$\frac{D^3}{2}$	1	2	$\frac{D}{\sqrt[3]{2}}$	$\sqrt[3]{2}$	0·1	1
$\frac{D^3}{4}$	2	4	$\frac{D}{\sqrt[3]{4}}$	$\sqrt[3]{4}$	0·2	2
$\frac{D^3}{8}$	3	8	$\frac{D}{\sqrt[3]{8}}$	$\sqrt[3]{8}=2$	0·3	3
$\frac{D^3}{16}$	4	16	$\frac{D}{\sqrt[3]{16}}$	$\sqrt[3]{16}$	0·4	4
$\frac{D^3}{32}$	5	32	$\frac{D}{\sqrt[3]{32}}$	$\sqrt[3]{32}$	0·5	5
$\frac{D^3}{64}$	6	64	$\frac{D}{\sqrt[3]{64}}$	$\sqrt[3]{64}=4$	0·6	6
$\frac{D^3}{128}$	7	128	$\frac{D}{\sqrt[3]{128}}$	$\sqrt[3]{128}$	0·7	7
$\frac{D^3}{256}$	8	256	$\frac{D}{\sqrt[3]{256}}$	$\sqrt[3]{256}$	0·8	8
$\frac{D^3}{512}$	9	512	$\frac{D}{\sqrt[3]{512}}$	$\sqrt[3]{512}=8$	0·9	9
$\frac{D^3}{n^3}$	Log n	n^3	$\frac{D}{n}$	n		Log n

Fig. 118.

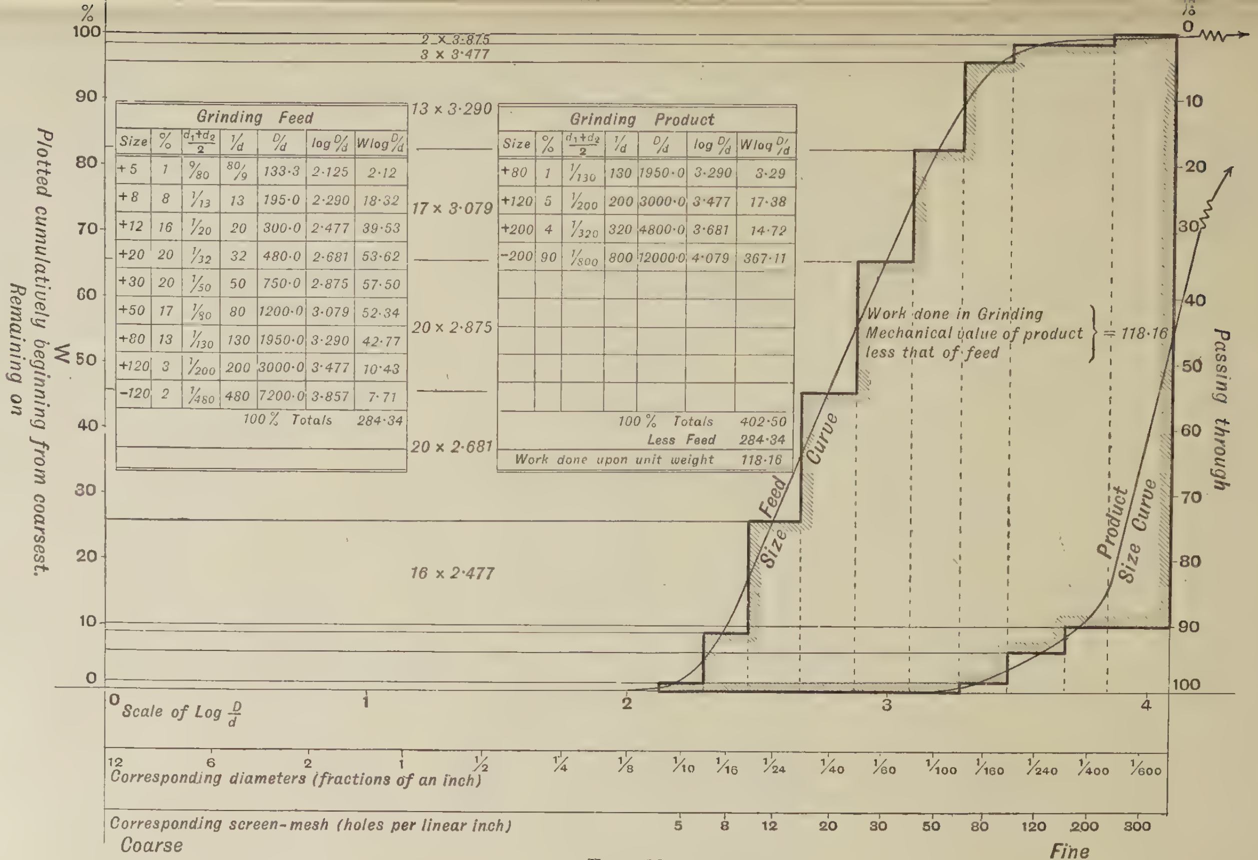
Work done in Crushing.—Tabular statement using as index the number of unit size-reductions (p. 187).

Summation of the work done in crushing a mass, D, to successive conditions of particle fineness, on the basis that unit volume-reduction represents unit work.

For convenience in illustration, unit volume-reduction has been taken to be the reduction to pieces each one half the volume of the preceding piece.

Applying these considerations to the determination of the work represented by a crushed mass made up of many sizes, the weights and diameters of these sizes are obtained and used as before (Fig. 119). The point of origin for the ordinates is again zero weight, while that for the abscissae is the zero logarithm obtaining where D/d is unity. D is conveniently taken to be that of the original material or any other larger diameter.

The divergence between these two methods of computing the work done in crushing is well shown by taking simple cases. Take, for instance, that of 16-in. pieces crushed to 1-in. pieces, which in turn are crushed to smaller pieces of $\frac{1}{16}$-in. dimension. Judged by the additional surface



Grinding Feed

Size	%	$\frac{d_1+d_2}{2}$	$1/d$	D/d	$\log D/d$	$W \log D/d$
+5	1	9/80	80/9	133·3	2·125	2·12
+8	8	1/13	13	195·0	2·290	18·32
+12	16	1/20	20	300·0	2·477	39·53
+20	20	1/32	32	480·0	2·681	53·62
+30	20	1/50	50	750·0	2·875	57·50
+50	17	1/80	80	1200·0	3·079	52·34
+80	13	1/130	130	1950·0	3·290	42·77
+120	3	1/200	200	3000·0	3·477	10·43
−120	2	1/480	480	7200·0	3·857	7·71
	100 %			Totals		284·34

Grinding Product

Size	%	$\frac{d_1+d_2}{2}$	$1/d$	D/d	$\log D/d$	$W \log D/d$
+80	1	1/130	130	1950·0	3·290	3·29
+120	5	1/200	200	3000·0	3·477	17·38
+200	4	1/320	320	4800·0	3·681	14·72
−200	90	1/800	800	12000·0	4·079	367·11
	100 %			Totals		402·50
				Less Feed		284·34
Work done upon unit weight						118·16

Fig. 119.

Representative Graph of Work done in grinding when measured by the Number of Unit Volume-Reductions.—Plot of calculations given on page 193. The area between the two stepped graphs of feed and product respectively, represents the work done. These two stepped graphs are smoothed into equivalent curves by joining the middle points of the vertical steps. The same curves would be obtained by a direct plot of the actual figures of the original sizing analyses (pp. 188, 199, 200, 214, Fig. 117).

exposed and assuming unit mass, the work done from step to step would be represented by $(1/d - 1/D)$, whereas by the number of volume-reductions suffered it would be Log D/d. The calculations would be as follows :

I. Additional surface exposed—

First step, 16″ to 1″; $(1/d - 1/D) = \frac{1}{1} - \frac{1}{16} = \frac{15}{16}$
Second step, 1″ to $\frac{1}{16}$″; $(1/d - 1/D) = 16 - 1 = 15$

Total work represented by $15\frac{15}{16}$
One step, 16″ to $\frac{1}{16}$″; $(1/d - 1/D) = 16 - \frac{1}{16} = 15\frac{15}{16}$

II. Number of particle volume-reductions—

First step, 16″ to 1″; $D/d = 16$; Log $16 = 1{\cdot}204$
Second step, 1″ to $\frac{1}{16}$″; $D/d = 16$; Log $16 = 1{\cdot}204$

Total work represented by $2{\cdot}408$
One step, 16″ to $\frac{1}{16}$″; $D/d = 256$; Log $256 = 2{\cdot}408$

In the above instance, judged by the additional surface exposed, the work done in the second step would be 16 times as much as that represented by the first step, whereas judging by the number of volume-reductions the two steps would represent equal work.

Further illustration of the incidence of these divergent expressions for work done in crushing, is well afforded by varying the above instance to the extent of making 2 in. the dimension of the intermediate pieces. The calculations would then be as follows :

I. Additional surface exposed—

First step, 16″ to 2″; $(1/d - 1/D) = \frac{1}{2} - \frac{1}{16} = \frac{7}{16}$
Second step, 2″ to $\frac{1}{16}$″; $(1/d - 1/D) = 16 - \frac{1}{2} = 15\frac{8}{16}$

Total work represented by $15\frac{15}{16}$
One step, 16″ to $\frac{1}{16}$″; $(1/d - 1/D) = 16 - \frac{1}{16} = 15\frac{15}{16}$

II. Number of particle volume-reductions—

First step, 16″ to 2″; $D/d = 8$; Log $8 = 0{\cdot}903$
Second step, 2″ to $\frac{1}{16}$″; $D/d = 32$; Log $32 = 1{\cdot}505$

Total work represented by $2{\cdot}408$
One step, 16″ to $\frac{1}{16}$″; $D/d = 256$; Log $256 = 2{\cdot}408$

This divergence may also be illustrated by calculations centred round mixed materials, in doing which the method of making such calculations is illustrated at the same time.

Thus, taking the materials disclosed by the following screen analyses to represent the feed and product respectively of the successive operations of breaking, crushing, and grinding :

REPRESENTATIVE SCREEN ANALYSES

	Breaking.		Crushing.		Grinding.	
Size.	Feed.	Product.	Feed.	Product.	Feed.	Product.
+ 12 in.	15					
+ 6 „	35					
+ 2 „	30	4	4			
+ 1 „	15	25	25			
+ 1/2 „	2	20	20			
+ 1/4 „	3	18	18			
+ 1/8 „		15	15			
+ 5 mesh		8	8	1	1	
+ 8 „		10	10	4	8	
+ 12 „				10	16	
+ 20 „				11	20	
+ 30 „				12	20	
+ 50 „				12	10	
+ 80 „				11	17	1
+ 120 „				9	13	5
+ 200 „				30	2	4
− 200 „						90
	100%	100%	100%	100%	100%	100%

MECHANICAL VALUE OF BREAKER FEED

By Surface Exposed. (Referred to infinite size.)					By Volume-reductions. (Referred to 15 in.)		
Size.	Per cent.	$\frac{d_1+d_2}{2}$.	$\frac{1}{d}$.	$\frac{W}{d}$.	$\frac{D}{d}$.	Log $\frac{D}{d}$.	W Log $\frac{D}{d}$.
+ 12 in.	15	15	1/15	1·00	1·0		
+ 6 „	35	9	1/9	3·88	1·7	0·227	7·94
+ 2 „	30	4	1/4	7·50	3·7	0·574	17·22
+ 1 „	15	$1\frac{1}{2}$	2/3	10·00	10·0	1·000	15·00
+ 1/2 „	2	3/4	4/3	2·66	20·0	1·301	2·60
− 1/2 „	3	1/4	4	12·00	60·0	1·778	5·33
	100	Totals .	. .	37·04			48·09

MECHANICAL VALUE OF BREAKER PRODUCT

By Surface Exposed. (Referred to infinite size.)					By Volume-reductions. (Referred to 15 in.)		
Size.	Per cent.	$\frac{d_1+d_2}{2}$.	$\frac{1}{d}$.	$\frac{W}{d}$.	$\frac{D}{d}$.	Log $\frac{D}{d}$.	W Log $\frac{D}{d}$.
+ 2 in.	4	4	1/4	1·00	3·7	0·574	2·29
+ 1 ,,	25	1½	2/3	16·66	10·0	1·000	25·00
+1/2 ,,	20	3/4	4/3	26·66	20·0	1·301	26·02
+1/4 ,,	18	3/8	8/3	48·00	40·0	1·602	28·83
+1/8 ,,	15	3/16	16/3	80·00	80·0	1·903	28·54
+ 5 mesh	8	9/80	80/9	71·11	133·3	2·125	17·00
− 5 ,,	10	1/20	20	200·00	300·0	2·477	24·77
	100	Totals . . .		443·43			152·45

WORK DONE IN BREAKING

	By Surface Exposed.	By Volume-reductions.
Mechanical value of product . .	443·43	152·45
Mechanical value of feed . . .	37·04	48·09
Work done on unit weight . . .	406·39	104·36

MECHANICAL VALUE OF CRUSHER PRODUCT

By Surface Exposed. (Referred to infinite size.)					By Volume-reductions. (Referred to 15 in.)		
Size.	Per cent.	$\frac{d_1+d_2}{2}$.	$\frac{1}{d}$.	$\frac{W}{d}$.	$\frac{D}{d}$.	Log $\frac{D}{d}$.	W Log $\frac{D}{d}$.
+ 5 mesh	1	9/80	80/9	8·88	133·3	2·125	2·12
+ 8 ,,	4	1/13	13	52·00	195·0	2·290	9·16
+ 12 ,,	10	1/20	20	200·00	300·0	2·477	24·77
+ 20 ,,	11	1/32	32	352·00	480·0	2·681	29·49
+ 30 ,,	12	1/50	50	600·00	750·0	2·875	34·50
+ 50 ,,	12	1/80	80	960·00	1200·0	3·079	36·94
+ 80 ,,	11	1/130	130	1,430·00	1950·0	3·290	36·19
+120 ,,	9	1/200	200	1,800·00	3000·0	3·477	31·29
−120 ,,	30	1/480	480	14,400·00	7200·0	3·857	115·71
	100	Totals . .		19,802·88			320·17

WORK DONE IN CRUSHING

	By Surface Exposed.	By Volume-reductions.
Mechanical value of product . .	19,802·88	320·17
Mechanical value of feed (for convenience to be the same as breaker product)	443·43	152·45
Work done on unit weight . .	19,259·45	167·72

Mechanical Value of Grinder Feed

By Surface Exposed. (Referred to infinite size.)					By Volume-reductions. (Referred to 15 in.)		
Size.	Per cent.	$\frac{d_1+d_2}{2}$.	$\frac{1}{d}$.	$\frac{W}{d}$.	$\frac{D}{d}$.	Log $\frac{D}{d}$.	W Log $\frac{D}{d}$.
+ 5 mesh	1	9/80	80/9	8·88	133·3	2·125	2·12
+ 8 ,,	8	1/13	13	104·00	195·0	2·290	18·32
+ 12 ,,	16	1/20	20	320·00	300·0	2·477	39·53
+ 20 ,,	20	1/32	32	640·00	480·0	2·681	53·62
+ 30 ,,	20	1/50	50	1000·00	750·0	2·875	57·50
+ 50 ,,	17	1/80	80	1360·00	1200·0	3·079	52·34
+ 80 ,,	13	1/130	130	1960·00	1950·0	3·290	42·77
+ 120 ,,	3	1/200	200	600·00	3000·0	3·477	10·43
− 120 ,,	2	1/480	480	960·00	7200·0	3·857	7·71
	100	Totals . . .		6952·88			284·34

Mechanical Value of Grinder Product

By Surface Exposed. (Referred to infinite size.)					By Volume-reductions. (Referred to 15 in.)		
Size.	Per cent.	$\frac{d_1+d_2}{2}$.	$\frac{1}{d}$.	$\frac{W}{d}$.	$\frac{D}{d}$.	Log $\frac{D}{d}$.	W Log $\frac{D}{d}$.
+ 80 mesh	1	1/130	130	130·00	1,950·0	3·290	3·29
+ 120 ,,	5	1/200	200	1,000·00	3,000·0	3·477	17·38
+ 200 ,,	4	1/320	320	1,280·00	4,800·0	3·681	14·72
− 200 ,,	90	1/800	800	72,000·00	12,000·0	4·079	367·11
	100	Totals . . .		74,410·00			402·50

Work done in Grinding

	By Surface Exposed.	By Volume-reductions.
Mechanical value of product . . .	74,410	402·50
Mechanical value of feed . . .	6,952	284·34
Work done on unit weight . . .	67,457	118·00

Assembling the figures representative of the work done upon unit weight at each stage, the following statement is obtained :

Relative figures of work done upon unit weight.

	By Surface Exposed.	By Volume-reductions.
In breaking . . .	406	104
In crushing . . .	19,259	167
In grinding . . .	67,457	118

Assuming that in breaking the capacity per h.p. hour is 2000 lb., in crushing 200 lb., and in grinding 100 lb., relative figures for the useful work done per h.p. hour in these respective operations, and consequently relative figures for the efficiencies of these operations in respect to power consumption, would be:

	By Surface Exposed.	By Volume-reductions.
Breaking . . .	812,780	208,820
Crushing . . .	3,851,890	33,544
Grinding . . .	6,745,700	11,800

There can be but little doubt that of these figures those arising from considerations of the number of particle volume-reductions express what is conceivably the truth, namely, that the finer the crushing the less efficient is the operation in the use of power. That must be so, even if only because the finer the particle the greater the difficulty of bringing force to bear, and the more often is the effort expended in vain. Surfaces which are true enough to seize coarser material will appear pitted and protective to fine material, suffering wear and consuming power themselves. Beam action can come into play with coarse material but not with fine.

It has to be remarked in conclusion that these theoretical considerations have led to no better design of crushing appliance. Calculation and diagram are, however, stimulating and a useful exercise, and there is no doubt that indirectly in the good which must follow careful examination of crushed products, they have their result.

CHAPTER V

SIZING ; LABORATORY SIZING

General Considerations

Sizing is the division of broken, crushed, or ground material, into classes according to size. In this connection size means average diameter, this being the ordinary conception and one readily appreciable by the eye. Large material, for instance, is spoken of as being of such and such a 'ring,' naming the diameter of the ring through which it will pass, while at the other end of the scale, magnification is likewise expressed in diameters. Neither cross-sectional area nor volume is so readily appreciated or compared.

The need for sizing lies in the fact that no one machine can satisfactorily handle material made up of a wide range of sizes. In comminution, sizing permits stage crushing : the lump ore to be broken, the broken ore to be crushed, the crushed ore to be ground. In separating the released mineral, sizing also permits separation in stages. If hand-picking is carried far it must be done in stages, since smalls are not adequately displayed in the presence of lumps. In water concentration, the means necessary to impress the required movement upon coarse material might carry away the fine. In magnetic separation, complete magnetization and removal of fine mineral might be accomplished while that of larger material remained imperfect. Summarizing, in dressing it often becomes necessary to eliminate variability in size in order that those mineral properties upon which dependence is placed to effect separation may have free play.

Neglecting sizing by air as practically inapplicable, two methods of sizing exist, namely, screen sizing, and water sizing or classification.

Screen sizing consists in permitting pieces or particles of ore to fall through openings larger than themselves, or in causing them to pass through such openings. Theoretically a particle should pass through an opening of the same cross-sectional area as itself, but in consequence of frictional resistance to passage, the particles which pass are smaller than the opening. Since also the average shape of crushed particles approaches the spherical or equidimensional, this resistance is greatest with round holes, less with square holes, and least with rectangular or slotted holes. Accordingly, where a slotted hole might be expected to pass, as maximum, a roughly spherical piece of a size equal to about 0·80*d*, *d* being the width of the slot, a square hole might be expected to pass a piece of 0·75*d*, *d* being the side of the square, and a round hole a piece of 0·70*d*, where *d* is the diameter of the hole. With respect to particles of other shape, the maximum of long pieces would by reason of the one long dimension be somewhat larger than that of round pieces; on the other hand, flat pieces having two relatively large dimensions would not be able to pass in such large average size as round pieces. Roughly, about 50 per cent of the particles of crushed material are equidimensional, 25 per cent long, and 25 per cent flat.

The simple products of screen sizing are an 'undersize' and an 'oversize.' Though sizing is designed primarily to eliminate one variable, that of size, it may, where the mineral is brittle, be also a means of separating an enriched undersize from an impoverished oversize (Fig. 161). It may indeed happen that the last stage in the concentration of the sandy portion of crushed material is a screening operation to separate an oversize poor enough for discard from an undersize worthy of further treatment.

Water sizing consists in dividing crushed material by submitting it to water rising at an appropriate velocity. Such sizing can only be applied to material fine enough to be borne in water; and it results in an 'overflow' of fine material and an 'underflow' of relatively coarse material. Where there exists no difference in density between the particles the result is pure sizing, the larger particle sinks and is separated below, the smaller particle rises and overflows with the water. But where difference in density exists the division is no longer entirely according to size but also according to density; a smaller particle will sink if it be of sufficient density, while a larger particle of less density might rise. The conditions might indeed be such that the underflow consisted in greater part of mineral, and the overflow in greater part of gangue; the operation would then have largely lost its character of sizing to become a separation by density (Fig. 207). On the other hand, when all the coarser mineral had been removed by previous concentrating operations, it might be found upon further water sizing that the overflow, carrying the friable mineral, was then the richer product.

Before dealing in greater detail with these two methods of sizing and the appliances whereby they are carried into effect, a description of the methods and procedure of making those sizing analyses which disclose the mechanical constitution of crushed material is opportune.

LABORATORY SIZING

Laboratory Screens.—Crushed material ordinarily ranges in size from about one-tenth of an inch down to impalpable powder. The division of such material into fractions each of determined particle-size, is accomplished by means of a series of woven-wire screens or sieves with square meshes or apertures defined by the dimension of the side, the several screens being successively finer.

The sequence of aperture in such a series is a question of some moment. It was natural to suggest that each succeeding size should be one-half the preceding size, that is, a ratio of 2 : 1 between consecutive sizes. Starting with an aperture of one-tenth inch and finishing with the finest wire-sieve possible of accurate manufacture, namely, one having two hundred holes per linear inch, there would be 6 screens in such a series, an insufficient number for close work; this series may be termed the simple or natural series. Rittinger in his time adopted the closer ratio of $\sqrt{2}$ or 1·414 : 1, equivalent to the reduction of the aperture area by one-half, obtaining thereby double the number of screens; this series is the square-root series. More recently Stadler and T. J. Hoover,[1] separately, proposed the ratio of $\sqrt[3]{2}$ or 1·259 : 1, equivalent to the reduction of the aperture-cube or particle-volume by one-half; this series, known as the cube-root series, gives about 18 screens within the above-mentioned range, which is perhaps more than sufficient: while, finally, Richards proposed the ratio of $\sqrt[4]{2}$ or 1·189 : 1, giving about 24 screens, a superabundant provision (Fig. 120).

Name.	Ratio.		Number of Screens between 5 mesh and 250 mesh.
	Numerical.	Descriptive.	
Simple . . .	2; 1	diameter halved	6 screens
Square-root . .	$\sqrt{2}$; 1·414	area halved	12 ,,
Cube-root . .	$\sqrt[3]{2}$; 1·26	volume halved	18 ,,
Fourth-root . .	$\sqrt[4]{2}$; 1·189	area quartered	24 ,,
Alternate cube-root	$\sqrt[3]{4}$; 1·585	...	9 ,,

FIG. 120.

Screen Series.—Tabular Statement.

These are all geometric series, each screen aperture having a fixed geometric relation to those above and below it, respectively. Except in the natural series, however, this relation, though simple of expression by symbols, is not simple of expression by figures, $\sqrt{2}$ being 1·414; $\sqrt[3]{2}$, 1·259; and $\sqrt[4]{2}$, 1·189, when taken to three places of decimals only. Starting then with a round number for the coarsest size, say 0·1 inch or 2·54 millimetres, the apertures of the following screens would largely be expressed by cumbrous fractions, vulgar or decimal, and in consequence the designation of these screens by their respective apertures has not been widely adopted.

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[1] *Trans. I.M.M.* Vol. XIX., 1910, pp. 471, 486.

Woven-wire screens are ordinarily designated by the number of meshes per linear unit, say, an inch, this number being known as the "screen mesh"; a 50-mesh screen, for instance, has 50 holes per linear inch. Since, however, an equal number of intervening wires alternate with these holes, the aperture of a 50-mesh screen is clearly not one-fiftieth of an inch, but more nearly one-hundredth, the exact dimension depending upon the diameter of the wire used. In consequence, though a 50-mesh screen would always have 50 holes to the inch, the aperture would be greater where a finer wire were used and less where the conditions necessitated a coarser wire. The conditions which determine the size of wire in commercial sizing do not, however, apply to laboratory sizing, and laboratory screens may therefore be made to suit their own particular requirements; the screens of finest mesh are indeed rarely used except in the laboratory.

Having in mind the simplicity of designation by mesh and realizing that the division of the material into a number of mathematically correct sizes was neither practically possible nor, if achieved, an end in itself, the Institution of Mining and Metallurgy, London, put forward a series of screens wherein, by making the diameter of the intervening wire equal to the aperture, mesh became standardized not only as the number of holes per inch but also as half the aperture reciprocal, that is to say, a 50-mesh screen would have 50 holes to the inch and an aperture of one-hundredth of an inch. In this series, with mesh in round figures the aperture in terms of the inch is equally in round figures, easily remembered and easily converted to any other unit, the millimetre for example.

With regard to the relation between consecutive screens, in this series no constant ratio exists, this constancy having been sacrificed to simplicity of aperture dimension and aptness of designation. With 17 screens in the complete series the ratio varies from 1·11 to 1·60, and on an average is 1·264, or approximately the same as the cube-root series. Whether in this divergence from a geometric ratio the sacrifice was disproportionate to the gain will largely appear from the following considerations: With a geometric ratio the range of size in any one fraction is the same as in any other. This regularity while scientifically gratifying is not the end in view, since in order to express in one summation the mechanical value of a mass of mixed particles, calculations based upon apertures must be made, these calculations giving a final figure in which any such regularity disappears; indeed, in obtaining this final figure the regularity of screen ratio has introduced cumbrous screen apertures. It is true that in plotting logarithms of aperture as abscissae, the successive screens of a regular geometric series find their places at regular intervals, which is not the case when the ratio is irregular (Figs. 119, 129). This, however, is no great advantage, since the proper position of any intermediate screen is readily determined from the logarithm of the aperture; moreover, in the final curve obtained, the regularity of screen ratio again disappears. It is of course obvious that too great a divergence from that regularity would endanger the trueness of the curve, but the small amount of latitude necessary to reap the advantages of simplicity of aperture and aptness of designation incurs no such danger. The I.M.M. series, given in the following table, is in consequence very convenient in the making of sizing analyses:

Screen Scale of the Institution of Mining and Metallurgy

Mesh.	Aperture.		Screen Ratio.
	Inch.	Millimetre.	
5	1/10	2·540	1·60
8	1/16	1·574	1·25
10	1/20	1·270	1·20
12	1/24	1·056	1·33
16	1/32	0·792	1·25
20	1/40	0·635	1·50
30	1/60	0·421	1·33
40	1/80	0·317	1·25
50	1/100	0·254	1·20
60	1/120	0·211	1·16
70	1/140	0·180	1·14
80	1/160	0·157	1·12
90	1/180	0·139	1·11
100	1/200	0·127	1·20
120	1/240	0·107	1·25
150	1/300	0·084	1·33
200	1/400	0·063	
		Average . .	1·264

From the I.M.M. series various convenient seriations may be selected. For instance, the series 5 mesh, 8 mesh, 12 mesh, 20 mesh, 30 mesh, 50 mesh, 80 mesh, 120 mesh, and 200 mesh, proposed by Stadler, is practically the geometric series having the screen ratio $\sqrt[3]{4}$ or 1·588. Moreover, it is a sequence easily remembered since the same figures, 5, 8, 12, 20, occur on either side of the central 30 mesh (Figs. 119, 129).

In the United States the square-root series based upon the aperture of a 200-mesh screen of particular make is largely used, the screens being known by their mesh. Mesh, however, in this series is only the number of holes per inch, and has no precise connection with the size of aperture. To obtain this latter, reference must be made to a table giving the measured apertures. It will be noted also from the table below that all these apertures are cumbrous decimals.

TYLER'S SCREEN SCALE

(Screen ratio $\sqrt{2}=1·414$.)

Mesh.	Aperture.	
	Inch.	Millimetre.
3	0·263	6·680
4	0·185	4·699
6	0·131	3·327
8	0·093	2·362
10	0·065	1·651
14	0·046	1·168
20	0·0328	0·833
28	0·0232	0·589
35	0·0164	0·417
48	0·0116	0·295
65	0·0082	0·208
100	0·0058	0·147
150	0·0041	0·104
200	0·0029	0·074

More recently the Bureau of Standards of the United States has recommended a screen scale based upon the millimetre, the screens above this base having a ratio of $\sqrt{2}$, or 1·414, and those below a ratio of $\sqrt[4]{2}$, or 1·189. Apertures of round-figure dimension occur with every alternate screen above the basal screen and with every fourth screen below. Round figures for the mesh, whether referred to the inch or the millimetre, being uncommon except in the very finest sizes, these screens are sometimes specified by their millimetric aperture.

SCREEN SCALE OF THE BUREAU OF STANDARDS, U.S.A.

(Screen ratio $\sqrt{2}$ above one millimetre, and $\sqrt[4]{2}$ below.)

Screen.	Aperture.		Mesh.	
Millimetre.	Millimetre.	Inch.	Metric.	Inches.
8	8·00	0·315	1·0	2·54
5·66	5·66	0·223	1·4	3·56
4	4·00	0·157	2·0	5·1
2·83	2·83	0·111	2·75	7·0
2	2·00	0·079	3·9	9·9
1·41	1·41	0·0555	5·0	12·7
1	1·00	0·0394	7·0	17·8
0·85	0·85	0·0335	8·0	20·3
0·71	0·71	0·0280	9·0	22·9
0·59	0·59	0·0232	10·0	25·4
0·5	0·50	0·0197	12·0	30·5
0·42	0·42	0·0165	14·0	35·6
0·36	0·36	0·0142	16·0	40·6
0·29	0·29	0·0114	20·0	50·8
0·25	0·25	0·0098	23·0	58·4
0·21	0·21	0·0083	27·0	68·6
0·17	0·17	0·0067	31·0	78·7
0·14	0·14	0·0055	39·0	99·1
0·125	0·125	0·0049	47·0	119·4
0·105	0·105	0·0041	59·0	149·9
0·088	0·088	0·0035	67·0	170·2
0·074	0·074	0·0029	79·0	200·7
0·062	0·062	0·0024	98·0	248·9
0·052	0·052	0·0021	110·0	279·4
0·044	0·044	0·0017	127·0	323·0

Conduct of Sizing Analyses.—The material being dry for convenience in weighing out the sample, the natural procedure is to conduct the operation dry. The amount of material taken will depend chiefly upon the extent of screen used, but also upon the number of fractions desired. The I.M.M. screens are made in squares of about 8 in., which it is usual to cut to a circle and surround with a stiff rim extending about two inches above the screen and three-quarters of an inch below (Fig. 121). This rim is so adjusted in diameter that the portion below fits inside the upper portion of the rim surrounding the next succeeding screen, permitting the screens to be nested in proper sequence, the coarse screen on top, the fine screen at the bottom. It is generally considered that this circular shape best forwards the swirling motion most effective in screening ; it undoubtedly avoids corners and the screen is convenient to handle and clean. By some, however, it is considered that these advantages are dearly bought at the sacrifice of screening area, and by these the original square shape is retained, when, in addition to the swirling motion, a rocking motion with one corner resting on the bench may be practised.

For wire screens of this size about 500 grm. of material is convenient. Assuming that the proportion of slime is not excessive, this amount is first screened on the coarsest screen, and the undersize in succession on the finer screens. Where, however, the amount of slime is great, the procedure is altered to take out as much as possible of the slime first, for two reasons, namely: because such an amount of slime requires the loosening effect of coarser particles to get it through the finest screen, and because if the coarse material were separated first it would be difficult to obtain it free from adhering slime.

In this dry screening by hand time is everything. Up to the limit where the attrition conceivably set up might vitiate the results, the longer the time given to each separate screening the more accurate the result. Extreme accuracy is, however, not necessary, but concordance between repeated results. The time for each screen must be set separately, since the passage of the undersize through the coarsest screen will take but a fraction of the time necessary for passage through the finest. The time should, in fact, increase regularly with the screen mesh and with the amount of material, and decrease with the screen area. With 1000 grm. on an 8-in. screen the mesh would fairly indicate the number of minutes necessary; with 500 grm. half that time would be sufficient. With screens as large as 18 in. as much as 2000 grm. may be used; on the other hand as little as 150 grm. has sometimes been found enough with small screens.

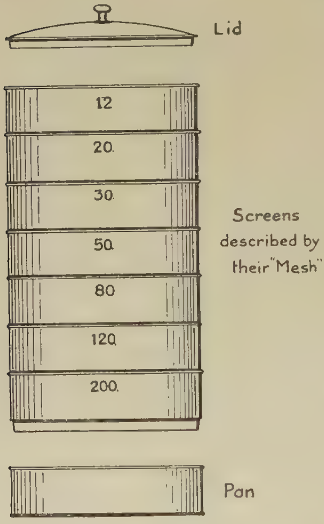

Fig. 121.

Nest of Screens.—Screens are usually framed in circular rims 8 inches in diameter; larger screens may be 10 inches. The sequence of screens indicated is one made up by selection from the I.M.M. series (pp. 199, 201).

If two screens are nested and worked together, as they may be if not too large and heavy, screening proceeds through both at the same time, and the time laid out as sufficient for the finer will more than suffice for the coarser. Could three or four be so nested and worked together the time taken for the whole nest would be but the time required for the finest. Such screening in nests can, however, only be done by a mechanical shaker; it cannot be maintained by hand. Nesting of the screens has a further advantage; it permits such a timely and regular arrival of the material upon each screen in succession that no packing occurs, but that which is fine enough finds practically unobstructed passage. Even by hand this beneficial effect is felt, to the extent that if a finer screen be used with the next coarser the passage through the former will take less than the time expected. When therefore several screens are nested and worked by machine there is a multiplied saving of time quite independent of the more regular and more rapid shaking; not only so, but there is no longer any need to take out the finest material first. It is not surprising, therefore, that with a mechanical shaker complete sizing analyses can be made in from five to twenty minutes. The shaking motion is variously gyrating, reciprocating, or rolling (Figs. 122, 123).

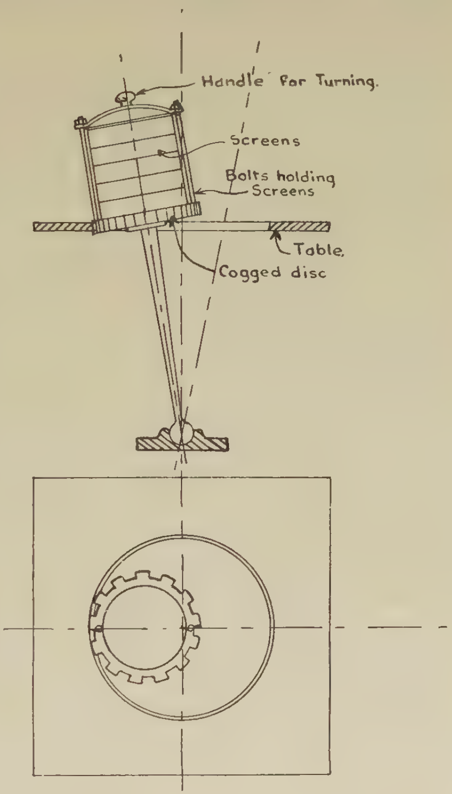

FIG. 122.

Laboratory Shaker.—The nest is fixed on a circular disc which itself is supported upon a central leg, standing up from a socket some two feet below a circular hole in a table. Around this hole the disc, and the nest with it, is gyrated. In this gyration the material rolls over as the disc continually changes the direction of its inclination, while at the same time square cogs around the periphery of the disc give rise to a continually-repeated knock (p. 203).

Another means of expediting these analyses is to place upon the screen some such body as an iron washer or small rubber ball, which, in responding to the movement, works in and through the material, keeping it from packing and the screen from blinding. Such a body is known as a beater. By its use the time taken to screen the finer sizes is practically halved; any objection that it causes grinding of the material is not serious.

Returning to the case of hand screening of material containing a good deal of impalpable slime, reasons were given why the bulk of this slime should be eliminated at once. This may of course be done dry, but is best done wet. So doing, the material is placed upon the finest screen under a stream of water, the fingers working it the while. By this water the fine material is carried through the screen into a receptacle beneath, where it is allowed to settle, is dried after draining off the water, and weighed. This washing of the sample to take out the finest portion may be more conveniently accomplished in a light sheet drum, with a suitable screen in the bottom and a handle across the top; or the sample may be washed in a bucket, the suspended fine material being poured off with the water. The oversize remaining on the screen or in the bucket is then dried, care being taken not to burst the particles by excessive heat, after which it is screened as when screening dry, this operation proceeding from the coarser to the finer. Arrived at the finest screen again, the further amount passing this screen is added to that previously carried through by the water. This mixture of dry and wet screening is described as the 'mixed method.'

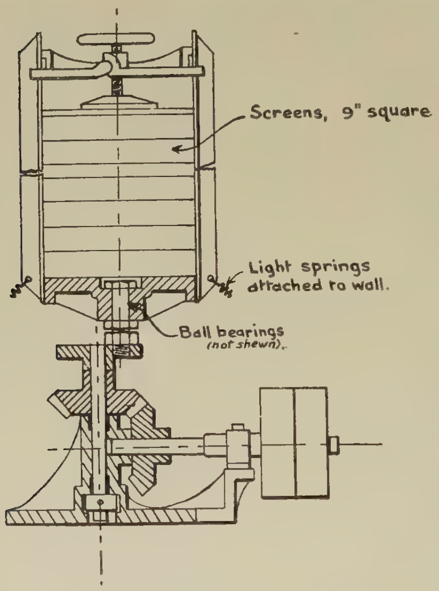

Fig. 123.

Laboratory Mechanical Shaker.[1] — In this shaker the motion is a gyration imparted in the same way as with the Coxe and Karlik screens. The sieves are square and the nest is kept from rotation by wire-springs attached to fixtures (pp. 203, 231).

When the assistance of water is extended to the coarse sizes also, the procedure is known as the 'wet method.' In this the screens are handled in the same sequence as with the mixed method, the slime being separated first, after which screening is started afresh, beginning with the coarsest. This wet method obviously necessitates the drying of all the sizes separately, which takes both time and attention, so that, though the actual screening occupies less time than the dry method, the total time taken is greater.

Comparing the three methods of dry, mixed, and wet screening, all by hand, it may be said that for material containing not too much slime the dry method is the most practical and rapid, while at the same time giving concordant results, that is to say, results agreeing within 1 per cent or so. It is the method most frequently employed. When, however, a good deal of slime is present dry screening takes longer and is less reliable, though fairly concordant results may still be obtained ; with such material the mixed method becomes the best : while, finally, with soft, friable material the complete wet method is probably the most reliable, giving results from which those from dry screening might differ widely. Probably, in any case the wet method though it takes longer is the most accurate, since with it there is little possibility of further attrition and no loss by dusting. With dry screening the loss by dusting and in manipulation will generally amount to about 1—2 per cent.

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[1] Hoover, *Trans. I.M.M.* Vol. XIX., 1910, p. 506.

Sometimes, when extreme care appears desirable, the products of wet screening are screened again dry, when any particles which may have been improperly compacted with water, become separated and conveyed to their proper fractions. Or, sometimes, wet screening may follow dry screening, to clean the larger particles of any adhering dust. Such extreme care is, however, of doubtful warranty since with wire screens the apertures vary in size, differences in area up to 30 per cent being of frequent occurrence. It may be remarked that silk screens, though rarely used, are more regular in aperture, and, because of the fineness of the thread, they also possess a greater discharge area and have a flatter surface, these properties facilitating discharge and lessening the chance of blinding.

Machine screening-analyses are made entirely and wholly dry. Six or seven screens are usually nested together in a column closed at the top by a cover and at the bottom by a pan. Being so closed there is a minimum loss by dusting ; in addition, the results are not only concordant but reliable. Seeing, therefore, that they take much less time, where many screening-analyses have to be made a mechanical shaker becomes a necessity.

Sizing Analysis by Water.—Fractionation by screening is limited by the inability to make accurate screens of exceedingly fine aperture, 200 mesh, or at most 300 mesh, being the present limit. In ordinary dressing there is luckily little need to go farther, but, in research and in the investigation of cyanidation residues, it is sometimes necessary to continue the analysis into the material passing the finest screen. This is accomplished by fall, relative or actual, in water, the larger particle falling quicker.

The fall of particles in a viscous medium—and with such small particles the viscosity of the water provides the discriminating resistance—was investigated by Stokes, who evolved the following formula for the hypothetical spherical particle—

$$v = \frac{2}{9} g \left(\frac{\delta - \delta_0}{n} \right) r^2 :$$

where δ = density of falling sphere,
δ_0 = density of viscous medium,
n = coefficient of viscosity,
g = gravity acceleration in centimetres = 981,
r = radius of sphere in centimetres,
v = velocity in centimetres per second.

Water at a temperature of 20° C. has a viscosity of 0·01, this viscosity increasing and decreasing about $2\frac{1}{2}$ per cent for every degree below and above that temperature, respectively. Inserting this value for viscosity, and unity for density of medium, and changing the radius r to diameter d, the formula becomes

$$v = 5450\ (\delta - 1)d^2,$$

or with conversion of the velocity and diameter into millimetres—

$$v = 545\ (\delta - 1)d^2.$$

Actual experiments by Richards confirmed this formula, giving a constant of 424 for quartz particles and 631 for galena.

Where no difference in density exists, which is the case being considered, $\delta - 1$ also is constant; making use of the figure determined by Richards and bringing all the constants together, the formula for the fall of minute quartz particles in water becomes—

$$v = 700d^2:$$

where v = velocity in mm. per second, and
d = diameter of the particle in mm.

With this simple relation between velocity of fall and diameter, it is easy to arrange a series of velocities to take the place of the impossible screens. For instance, 300 mesh would correspond to an aperture of $\frac{25}{600}$ mm., through which the maximum size of particle which might be expected to pass, quartz breaking in roundish particles, would have a dimension roughly 80 per cent of the aperture, or $\frac{20}{600}$ mm. For the fractions to be obtained by velocity to be properly continuous with those obtained by screening this diminished diameter is required. Using it, the velocity becomes about 0·8 mm. per second, though that of a particle having the full size of the 300-mesh aperture would be 1·21 mm. In the same way the respective velocities for 500 mesh would be 0·28 mm. and 0·44 mm., and those for 800 mesh, 0·11 and 0·17 mm. per second (Fig. 124).

This division into fractions by the varying velocity of fall of different-sized particles may be accomplished either directly by fall in still water, this method being known as "sedimentation," or by submitting the particles to rising currents of calculated velocity, this being known as "elutriation."

Mesh.	Aperture.		Velocity per second.	
	Inch.	Mm.	Full size.	80 per cent Full size.
200	$\frac{1}{400}$	0·0635	2·73 mm.	1·75 mm.
300	$\frac{1}{600}$	0·0423	1·21 „	0·80 „
500	$\frac{1}{1000}$	0·0254	0·44 „	0·28 „
800	$\frac{1}{1600}$	0·0156	0·17 „	0·11 „
1200	$\frac{1}{2400}$	0·0110	0·07 „	0·045 „

Fig. 124.

Elutriation Velocities of Quartz Particles.—Velocities corresponding to particular screen apertures; calculated from Stokes' formula using Richards' constants (pp. 206, 276).

Sedimentation is the more simple but less accurate method. A weighed amount of material, say 500 grm., is placed with water in an upright tinned or glass cylinder of about 120 mm. diameter and 300 mm. height, the water reaching to a mark near the top. In the side of this receptacle, at depths dependent upon the products it is desired to draw off, are plugged discharges (Fig. 125). The cylinder is shaken till all the material is in suspension, when, at a given moment, such agitation is stopped. Then after a set interval the upper discharge is opened and the pulp above it drawn off and set apart. This set interval is calculated as if all the particles to be separated started to fall from the water surface. The particles, on the contrary, being equally distributed through the water, those which start below the level of discharge as well as those from points not high enough above, have no opportunity to pass out as by their size they should, but, settling down, remain in the cylinder. The operation has accordingly to be repeated till it is considered that these particles, or at least the great bulk of them, have been discharged. To ensure this, about five separate operations are generally necessary, the separate discharges being all put together, settled, decanted, dried, and weighed.

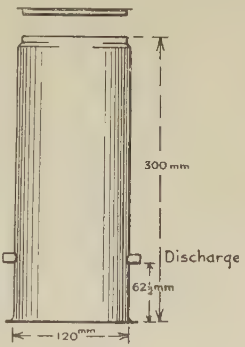

Fig. 125.

Sedimentation Vessel.—A tinned cylinder with a lid fixed by being pressed firmly into place, is illustrated. Into its side two tubes for discharge are soldered one on either side. Different products are obtained through this single discharge level, either by varying the depth of water placed in the vessel, or by varying the time of settlement (p. 207).

The same procedure, but varied either by adopting a shorter time for the same discharge or by keeping the same time-interval for a deeper discharge, will deliver a second and coarser product if such is desired. Thus :

Assume that it be desired by ten minutes' settlement to separate material finer than 800 mesh. The falling velocity of the maximum quartz particle which would pass a screen of that aperture, if such could be made, would be 0·11 mm. per second. At that rate it would fall 66 mm. in ten minutes, and a discharge set at that depth below the water surface would make the desired separation. Similarly, the particle passing a 500-mesh screen would have a falling velocity of 0·28 mm. per second or 168 mm. in ten minutes, and a discharge at that depth would be effective. For the 300 mesh the interval of ten minutes would be too long, since the particle passing that screen would fall 0·80 mm. per second ; taking advantage of the discharge at 168 mm., such a particle would be there discharged after a time-interval of three minutes. Finally, the particle passing 200 mesh would have a falling velocity of 1·75 mm. per second, at which rate it would take roughly 100 seconds to reach the lower of the two discharges. After these four fractions had thus been separated, the material remaining would all be larger than 200 mesh and in excellent condition for dry screening.

It is rarely, however, that such an extended separation is performed by sedimentation, since elutriation is more accurate and more conveniently undertaken. Sedimentation is very appropriately employed when it is desired to separate the material smaller than 200 mesh into two portions only, namely, fine sand and impalpable slime. For this purpose the velocity corresponding to 300 mesh would be proper.

Where accurate and complete fractionation is necessary elutriation must be employed. Elutriation is based on the fact that a particle will just be sustained in an upward-rising current of water if the velocity of that current be equal to that which the particle itself would attain when falling in still water ; a greater velocity would carry the particle upward, a smaller velocity would allow it to drop. Accordingly, by placing the material in a tubular vessel up which water is rising at a velocity under control by the quantity permitted to pass, the velocity may be so adjusted that particles finer than a given size will be carried upward to the overflow while those coarser will sink.

If only one tube is used the finest material is separated first by employing the lowest velocity. When that is well away, the next product is carried over by the application of a greater velocity, and so on, until all has gone over or till nothing remains but such coarse material as can best be screened. The Stadler-Schoene elutriator[1] is of this type (Fig. 126). The tube or cylinder, about 400 mm. long and 40 mm. diameter, is closed on top by a cork, while at the bottom its diameter gradually contracts till about the size of ordinary glass tubing, when it is bent upwards to be connected by rubber tubing to the tap of a steady-head reservoir above. The quantity of water passing, and consequently the velocity in the cylindrical vessel, is under control by this tap.

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[1] *Trans. I.M.M.* Vol. XXII., 1913, p. 686.

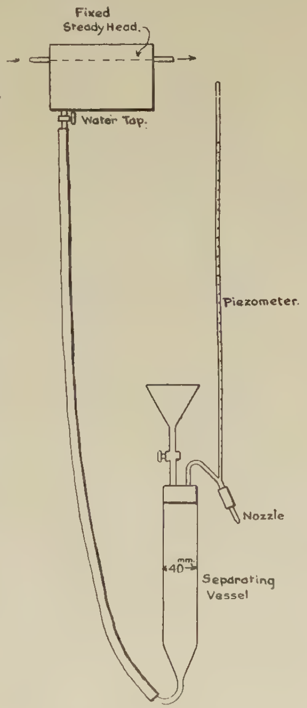

FIG. 126.

Stadler - Schoene Elutriator. — Glass Vessels (p. 208).

The overflow from the vessel is upwards through a glass tube inserted in the cork, the actual discharge taking place through a nozzle attached by a short piece of rubber tubing, the whole arrangement resembling very much the discharge from the familiar wash-bottle. It is distinguished, however, by having an upward branch just before the nozzle, this branch being a graduated vertical tube some three feet in length, serving as a piezometer. Discharge being limited by the size of the nozzle, the water rises in this piezometer till the increasing pressure of its height forces the discharge of all the water which the tap permits to enter. Measuring the amount of water discharged under different heights, and knowing the cross area of the separating vessel, the piezometer can be calibrated in terms of rising velocity; it accordingly constitutes a quick and accurate means of obtaining by tap-adjustments any desired succession of velocities. With change in the size of nozzle the piezometer must be recalibrated.

The material submitted to the operation of this elutriator, generally about 50 grm. dry weight, is charged with water through the cork by means of a stoppered funnel. The lowest velocity is first employed, the equivalent water-quantity of which is small. The elimination of the finest material accordingly takes a long time—sometimes as much as two days—an inconvenience accentuated by the opportunity taken by the larger material to pack itself into the conical bottom, withdrawing thereby some of the fine material from the searching action of the rising water, the water not passing through the pack. To minimize this packing some coarser rounded grains or shot are best added, while to ensure that no fine material is locked together in floccules the addition of a deflocculating agent, sodium carbonate for instance, is advisable. Afterwards, by the employment of greater velocities the successively larger products are carried over relatively quickly, settled, dried, and weighed. Since, however, these have to wait their turn, a complete sizing analysis of fine material takes a long time, most of which is occupied in expelling the finest size.

To cut short this time, elutriators with two or three separating vessels in series are favoured. This compounding of vessels has parallel advantages to the nesting of the screens; it not only saves time by separating the sizes concurrently, but also in eliminating the opportunity for the material to pack. Fed at the top into the slow-velocity vessel, the material is at once divided, the coarser portion dropping into its own particular vessel where it is tossed by a high velocity and cannot pack. Such a disposition of separating vessels in vertical series is employed in the Crook elutriator (Fig. 127). This elutriator, which has been used chiefly in the analysis of natural sediments, consists of two separating vessels, the upper of 90 mm. diameter tapering to 15 mm., and the lower of 15 mm. diameter, the two being joined by rubber tubing. From the bottom of this lower vessel a length of rubber tubing leads upward to an adjustable steady-head—a small glass contrivance clamped to a vertical rod upon which it can be raised or lowered—the vertical position of which determines the precise amount of water passing. After calibration, the relevant velocities are indicated on a piezometer rising through the cork closing the upper vessel, through which cork the discharge tube also passes. When separating sand in the narrow vessel, silt in the wider vessel, and clay in the discharge, the quantity of water passing through the above vessels would be 100 c.c. in 90 seconds. The amount of dry material taken is about 10—20 grammes.

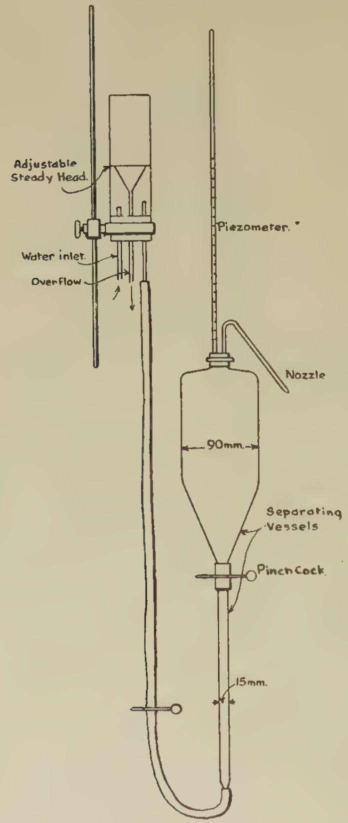

Fig. 127.

Crook Elutriator.—Glass Vessels (p. 210).

In the examination of crushed products, water sizing would continue the series begun by the screens, that is, would separate particles respectively smaller than, say, 800 mesh, 500 mesh, and 300 mesh; and accordingly, the material submitted to elutriation would be that passing 200 mesh. The velocity in the narrow tube would then be arranged to carry upward the material less than 300 mesh, and that in the larger tube to lift and discharge material less than 500 mesh. Using the quantity of water given above, namely, 100 c.c. in 90 seconds, and accepting the velocities at 0·80 mm. per second and 0·28 mm. respectively, proper diameters for the tubes would be 42 mm. and 71 mm. respectively. Using two tubes only, there would be no opportunity to divide further the material smaller than 500 mesh, but by adding a third vessel of 113 mm. diameter a fraction smaller than 800 mesh could be made. Other quantities of water would require other diameters.

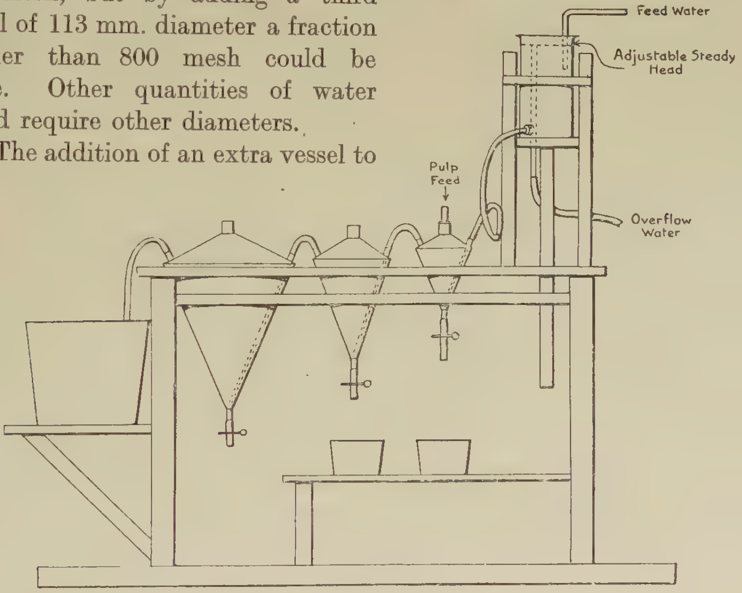

Fig. 128.

Nipissing Elutriator.—Tinned Vessels (p. 212).

The addition of an extra vessel to the Crook elutriator would, however, give it an inconvenient height, and elutriators with the three vessels side by side in series, the top of one being connected to the bottom of the next downstream, are better. In such case the material to be elutriated is entered through the top of the high-velocity vessel, from whence the finer material is driven downstream to find its place in one of the succeeding vessels, or, if of the finest size, in the discharge. An elutriator used by Beringer in the examination of tin concentrate [1] was of this type.

For accurate determinations the separating vessels must have sufficient length, seeing that they not only separate but contain the material, much of which is loosely balanced in the stream. Where, however, a high degree of accuracy is unwarranted, shorter vessels with conical bottoms and corked tops are used, the elutriator becoming a series of small classifiers. For the better control of crushing at the Nipissing mill at Cobalt, Canada, such a type of elutriator with metal cones in succession $5\frac{1}{2}$ in., 7 in., and $10\frac{1}{2}$ in. diameter was successfully employed[1] (Fig. 128).

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[1] *Trans. I.M.M.* Vol. XXIV., 1915, p. 416.

Records of Sizing Analyses.—The screens and velocities used divide the material into fractions, each fraction lying between two succeeding screens or velocities. These products are described by placing a *minus* sign before the mesh of the screen through which the material has passed, and a *plus* sign before that on which it remains. Fractions obtained by elutriation may equally be described in terms of the equivalent limiting meshes. A complete sizing analysis would then be recorded in three columns: the first a *minus* column of the screens through which the undersize passed in succession, the second a *plus* column of the screens on which in succession some oversize remained, and the third a column giving the weight of each fraction as 'simple percentage' of the total weight, thus:

SIZING ANALYSIS

Mesh of Limiting Screens.			Simple Percentage of Total Weight.
− 20	+ 30	=	15
− 30	+ 50	=	20
− 50	+ 80	=	25
− 80	+120	=	22
−120	+ ∞	=	18
− 20	+ ∞	Total .	. 100

If one or other of the screen columns be suppressed and the analysis be recorded either in terms of the percentages passing through, or of percentages remaining on the several screens in succession, the simple percentages of weight must at the same time give place to 'cumulative percentages,' thus:

Screen Mesh.		Passing through. Cumulative Percentage.
− 20	=	100
− 30	=	85
− 50	=	65
− 80	=	40
−120	=	18

Screen Mesh.		Remaining on. Cumulative Percentage.
+ 30	=	15
+ 50	=	35
+ 80	=	60
+120	=	82
+ ∞	=	100

This numerical record, though clear, complete, and necessary for calculation, yields to graphic representation when it is desired to convey the mechanical condition of a mass of mixed particles at a glance (Fig. 129).

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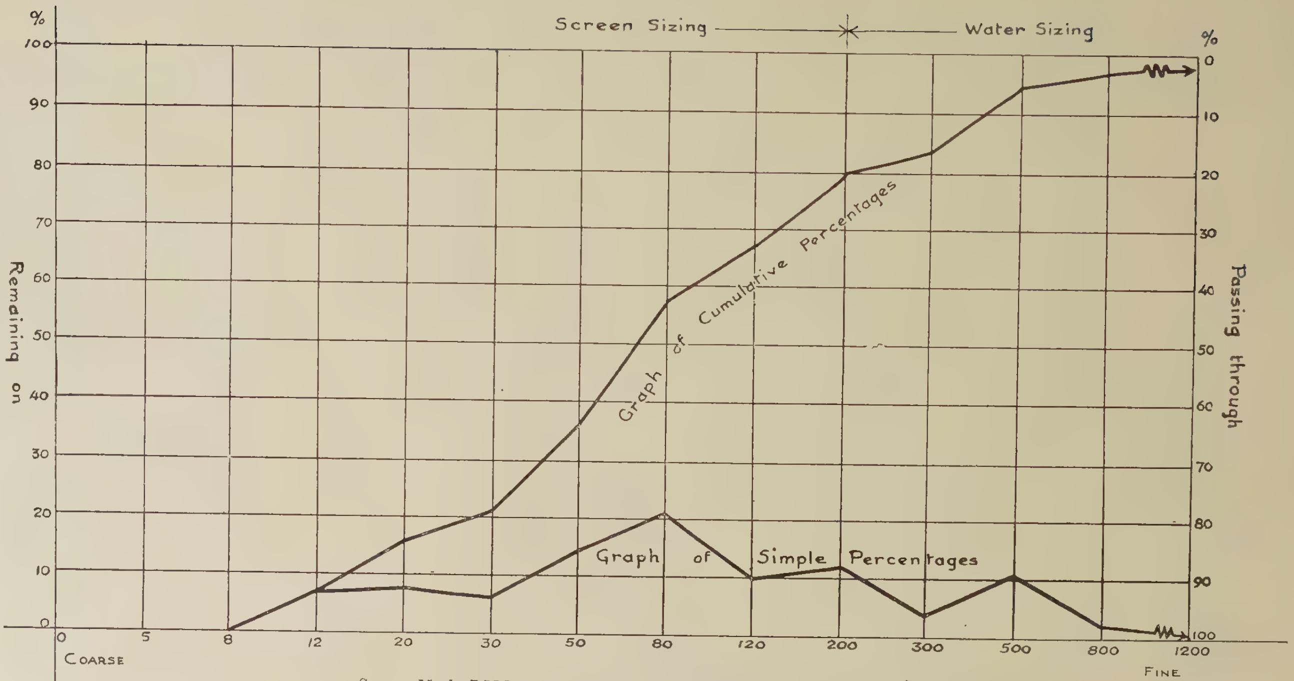
[1] *Trans. Can. Min. Inst.* Vol. XVII.. 1914, p. 83.



Screen Mesh, I.M.M. Standard. Aperture-ratio, rough average, $\sqrt[3]{4}=1{\cdot}588$.

FIG. 129.

Plot of Sizing Analyses.—Analyses made with screens having a geometric aperture-ratio (pp. 186, 199, 200, 212, 214).

Such graphic representation, as already indicated, is best provided by plotting the cumulative percentages of weight as ordinates, and the logarithms either of the aperture, of the diameter-reciprocal, of the screen-mesh, or of the size-reduction as abscissae—such logarithms giving equality to the same amount of size-reduction wherever it occurs. For instance, the reduction of a mass of particles from 120 mesh or 0·17 mm. aperture to particles of 200 mesh or 0·063 mm. is given the same importance in the diagram as that, say, in reducing a like mass from 5 mesh or 2·54 mm. aperture to 8 mesh or 1·57 mm. aperture, the size-reduction in each case being the same (Fig. 119). Moreover by such a logarithmic plot the diagram is open at either end and neither the coarser nor the finer sizes are cramped (pp. 185, 186).

If, instead of the logarithmic functions, the reciprocals of diameter or their equivalents be plotted, the finer sizes occupy a disproportionately large portion of the diagram, while the coarser sizes are cramped, zero appearing as the reciprocal of an infinitely large size. Moreover, a given size-reduction in the finer stages would present itself very much larger than the same amount of size-reduction in the coarser stages (Fig. 117).

Or, if the actual apertures themselves be plotted—which is more rarely done—zero appears as the infinitely small aperture from which the others are measured. The large apertures then occupy far too much of the diagram, while the smaller apertures are cramped and make no satisfactory display.

Another advantage of a plotted diagram lies in the fact that quantities lying between any two screens other than those taken, may be directly measured. In this respect the logarithmic plot has again an advantage over that of the simple functions, in that the measure so taken gives effect to the view that the mean diameter of particles between two apertures is properly the logarithmic mean of those two apertures and not the arithmetic mean.

The logarithmic plot shows its superiority again, when, instead of the cumulative, the simple percentages are the ordinates. The graph so resulting is not a continuous curve but a jagged line, rising with increase in percentage and falling with decrease (Fig. 129). The high points in such a line indicate clearly of what size the bulk of the material is, a piece of information most useful when determining the type and number of the machines to follow. As will be realized later, such a graph is of service when controlling the work of classifiers. In its construction there is no reason why either the coarse end or the fine should be cramped, as would happen if simple functions were plotted. Equality in display is all that is demanded and this the logarithmic plotting alone affords.

Finally, the maximum of information and the completest picture are afforded when both simple and cumulative percentages are plotted on the same graph (Fig. 129).

Microscopic Measurement of Size.—Measurement of particle size by the microscope is best performed by using an eyepiece micrometer throwing a graticule scale into the field. A stage micrometer may be used to check the correctness of the eyepiece scale, or to determine the value of that scale when a different objective is used. For dressing purposes a magnification of 40—50 diameters is generally sufficient, while it possesses the advantage of giving a flat field; an actual field of 2—2·5 mm. then appears about 100 mm.[1] The unit of measurement for the finest material is the *micron* or 0·001 mm., often symbolized as μ. Each division on the micrometer scale should represent 25 or 12·5 of these units. An intermediate magnification, say, of 150 diameters would permit approximations to be made of much smaller sizes, though such a magnification is somewhat unsatisfactory because of difficulties of focus. A high power of, say, 450 diameters would only be used in investigations of material like clay. Moir, in an interesting paper on the measurement of particles of dust in mine air,[2] has shown that a good idea of the size of particles of about one micron can be obtained by using a magnification of more than 1000 when, to help estimation, the field is divided by a graticule scale into squares of known size.

Direct measurement of particles under the microscope down to a minumum size of about 0·025 mm. may be effected when the microscope is provided with a special sliding platform, by arranging the particle tangential to a cross wire in the field and then sliding the platform till the diameter of the particle has been traversed, the amount of this movement being read by a vernier scale.

Particle measurement by the microscope, though the only means of actually measuring minute particles, is subject to the error that as the particles lie flat in the field the largest diameter is presented and of course measured.

SIZE-NOMENCLATURE OF PRODUCTS

The largest size of crude ore is known as 'lump' ore or 'lumps,' the lower limit of which may be put at 6 in. Then come 'cobbles,' 1½ in. or 2 in.; and 'smalls,' the material smaller than cobbles. Of crushed ore, that above 5 mesh is generally described by its size as 1 in., ½ in., ¼ in. material, etc.; from that to 30 mesh is 'coarse sand'; while from coarse sand to 200 mesh is 'fine sand.' Of the material finer than 200 mesh the granular portion is 'very fine sand,' and the impalpable portion 'slime,' though often the whole of it is termed slime.

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[1] Beringer; *Trans. I.M.M.* Vol. XXIV., 1915, p. 439.

[2] *Journ. Chem. Met. and Min. Soc. of South Africa,* Vol. XVI., 1915–16, p. 1.

CRUDE ORE

Lumps	Pieces greater than 6 inches.
Cobbles	Pieces between 6 inches and 2 inches.
Smalls	Pieces smaller than 2 inches.

CRUSHED ORE

Crushed smalls, or Roughs	Pieces above 5 mesh.
Coarse sand	Particles between 5 mesh and 30 mesh.
Fine sand	Particles between 30 mesh and 200 mesh.
Very fine sand	Granular material smaller than 200 mesh.
Slime	Impalpable material.

In the technology of natural sediments that described above as slime is clay, and the terms sand, silt, and clay have more precise meanings. Hatch and Restrall[1] give the following size-definition to these terms :

Coarse sand	0·75 mm.—2·50 mm.
Sand	0·10 mm.—0·75 mm.
Fine sand	0·05 mm.—0·10 mm.
Dust (or mud)	0·00 mm.—0·05 mm.

Crook, in an appendix to the work of the authors just quoted, gives :

Sand	0·10 mm.—1·00 mm.
Silt	0·01 mm.—0·10 mm.
Mud	0·00 mm.—0·01 mm.

While Boswell, in a Memoir published by Ministry of Munitions,[2] gives :

Very coarse sand	1·00 mm.—2·00 mm.
Coarse sand	0·50 mm.—1·00 mm.
Medium sand	0·25 mm.—0·50 mm.
Fine sand	0·10 mm.—0·25 mm.
Superfine sand	0·05 mm.—0·10 mm.
Silt	0·01 mm.—0·05 mm.
Clay or mud	0·00 mm.—0·01 mm.

In ore-dressing, descriptions have to suit the products into which it is found convenient to divide the crushed material ; it is not surprising that these products being artificial do not conveniently fall into the same divisions as the natural sediments. Natural sediments indeed differ among themselves, and it is more than likely, for instance, that a technologist devising descriptions for the fractions of oil-sand would arrive at a different division from that found suitable for moulding- or glass-sands.

When ore is crushed and ground in water, as most of it is, some of the resultant slime, though still visible to the microscope, remains more or less permanently suspended. Such a system of medium and particle constitutes a 'suspension,' the solid material being a 'suspensoid.' Finer still, some of the material is only detectable by the ultra-microscope; such a combination constitutes a 'colloidal solution,' the solid material being a 'colloid.' Finally, it is doubtless the case that an insignificant portion of the material passes into true solution where the particles, still smaller again, are of molecular or ionic size (p. 510).

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[1] *The Petrology of the Sedimentary Rocks*, London, 1913.

[2] *Sands suitable for Glass-Making.*

To give precision to this conception of the three conditions, Zsigmondy[1] proposed the following size-description:

Suspensions	Particles greater than 0·1 micron.
Colloidal solution . . .	Particles between 0·1 and 0·001 micron.
True solutions . . .	Particles less than 0·001 micron.

Between these three conditions or systems there exists perfect continuity in size, the marked differences in properties being largely those of degree. In a coarse-sand suspension, gravity is in control and the particles settle. With decreasing size gravity loses force; and, finally, molecular forces assume control. The particle then remains in more or less permanent suspension.

In ore-dressing the troublesome properties of colloidal slime are slow settlement and low permeability after settlement, troubles which increase as the particle-diameter decreases. Small size of the particle accounts for this trouble, though the participation of surface alteration of the particle is not excluded. That this connection with size has not been more obvious is because the trouble encountered bears no close relation either to the amount of grinding, or to the amount of fine material disclosed by sizing analyses conducted only with screens. The trouble depends upon the extent to which the ore of itself goes to infinite division under grinding, an extent not disclosed by such analyses. Two materials having the same proportion passing the finest mesh, the one consisting of very fine sand and the other of impalpable slime, would behave very differently. Or again, material which from its known nature must upon crushing go largely into impalpable slime might be found to present no difficulty; such might very well happen when an undesigned flocculation was brought about by electrolytes unsuspected in the mill water. The amount of impalpable slime accordingly depends primarily upon the character of the ore itself, upon its previous history, the changes it has undergone. Weathered ore, for instance, produces colloidal slime because weathering produces easily-disintegrated minerals. Crushed sand, on the other hand, washed free from any associated colloidal slime does not upon further grinding again yield any appreciable amount of such material. Ordinary grinding does not of itself give rise to colloidal slime; it does, however, release colloidal material present in the ore.

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[1] *Erkenntnis der Koloide*, p. 22.

CHAPTER VI

SIZING ; SCREEN SIZING

Screens

In practice, the openings or apertures through which the undersize is given opportunity to pass may be either the spaces between steel bars placed parallel to one another, such an arrangement constituting a " bar screen " ; or they may be the holes, round, square, or slotted, punched in metal plate, such an arrangement constituting a " punched-plate screen " ; or, finally, they may be the square or rectangular holes made by weaving two sets of fine metal wires at right angles, this arrangement constituting a " woven-wire screen."

The bar screen is used for coarse material or where the screen is subject to heavy wear. It generally consists of wedge-shaped steel bars set on end alongside one another with regular spaces between—1½ to 4 in.—these bars being held in place either by bolts through them or by clamps at their ends (Fig. 132). Other sections than the wedge shape may be used ; ordinary steel rails turned bottom upwards are, for instance, found convenient and suitable with very large material, while plain rectangular bars, and even round bars, occasionally find application.

Punched-plate screens are used for an intermediate size of material. With them the holes, which range in diameter or width from about 1½ in. to 30 mesh, may be round, square, or slotted, holes of this last shape generally having rounded ends (Fig. 130). The round hole is that most commonly used, then the slotted hole, while the square hole is uncommon. In punching these holes two methods are followed : either a wad is punched clean out of the plate, as with the larger holes, a clear-punched hole resulting ; or the plate is merely pierced, as with holes less than a millimetre, the punch being then no longer capable of cutting a clean wad. In thus piercing the plate the punch at its exit breaks the smoothness of that side, a burr-punched hole resulting. It is this burred side which becomes the working side of the screen ; the hole there being of smaller diameter anything entering will be assured of passage, and blinding of the screen is minimized.

The closeness at which the holes are placed to one another varies with the size of the hole. Larger holes permit as well as demand greater thickness of plate, and with this greater thickness the web between the holes

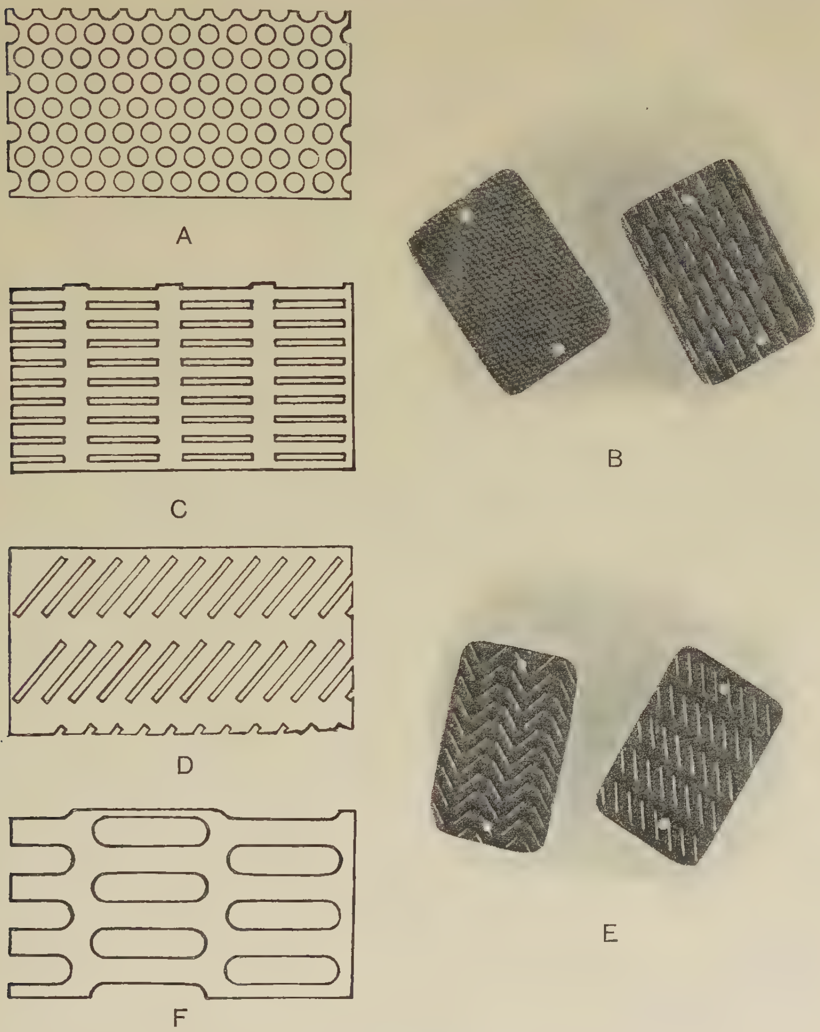

Fig. 130.

Punched-plate Screens.—A, round perforations along lines at 60° ; holes about half-diameter apart. B, burr-punched round perforations along lines at 60° ; holes about two and a half diameters apart. Also, slotted perforations, lying horizontally and staggered to break the vertical line. C, horizontal slots, in line. D, diagonal slots with horizontal webs. E, diagonal slots with vertical webs. Also, alternately-sloping diagonal slots. F, oblong slots, staggered endwise (pp. 218-220).

may be narrower and yet be strong enough. With steel plate and holes not less than $\frac{1}{4}$ in. diameter or width, the web between the holes is, more or less, equal to half the dimension. When, however, the holes are very small the plate is very thin, and in order that the web may be strong enough the holes must be farther apart, in the finer sizes two or more diameters. Where brass, bronze, or copper screens are used, as when the water is acid or contains copper, a greater distance between the holes is necessary on account of the lower strengths of these materials. Punched screens are not ordinarily made below 30 mesh, the punch itself being then too weak for the work.

The diameter or width and the minimum distance between the holes having been fixed, there still remains the choice of their arrangement. Round holes may either be arranged along lines at right angles or along lines making 60° with one another. Of these two arrangements the latter is preferable, since in the same area it gives a greater number of holes, that is to say, it gives a greater discharge-area. Expressing this area of discharge as a percentage of the total area—the usual way of making the comparison—the coarser screens with holes at half diameter apart will possess a discharge area of 40 per cent when the holes are along lines at 60°, but only 35 per cent when along lines at right angles. With finer screens and holes a full diameter apart the figures would be 22·5 per cent and 20 per cent respectively (Fig. 130).

Slotted holes permit a variety of arrangement. The slots may be normal and in line, or normal and staggered; diagonal and in line, or diagonal and staggered. The normal slots may be vertical or horizontal; the diagonal slots may all slope one way, which is more common, or they may slope alternately one way and the other. Staggering may be to break either the vertical or the horizontal line (Fig. 130).

Woven screens are ordinarily employed for material from $\frac{1}{4}$-in. diameter down to the finest size, though exceptionally also for larger material. They are made by weaving two sets of wires at right angles (Fig. 131); with one set of straight wires—the warp—crossed and kept in place by a set of crimped wires—the woof—the screen is single-crimped; when, however, both sets are crimped, a double-crimped screen results, the wires of which are better locked and the aperture in consequence more regular. This latter type is that generally used.

Since with woven screens the wires run alternately under and over one another, the surface is not smooth, and the freedom of the particles to move upon it, suffers. Moreover, the aperture is a little sunken and tilted, giving the screen a tendency to hold the particles and thereby to become blinded. To remove such defects these screens are often passed between rollers, the resultant ‘rolled screens,’ presenting their apertures more normally to the particles, being quite satisfactory (Fig. 131).

Concerning the relation of the aperture dimension to the diameter of the wire, with screens of ordinary aperture the wire is always the smaller, and yet sufficiently strong; but with very fine screens the wire may be the greater. The percentage of discharge area naturally depends upon the relation of these two dimensions. With square apertures this area varies from about 50 per cent in the coarser woven-screens to about 33 per cent in the finer. Compared with punched screens of the same aperture the discharge area is considerably greater. In practice, also, woven screens ensure a more constant aperture than punched screens, partly because as the burr wears the effective aperture of the latter increases, but also because woven screens have a shorter life, their end coming by wire breakage, while punched screens generally remain in use till the holes have worn too large. A further point in favour of woven screens is their lower weight and price; a point against them is their somewhat greater tendency to blind.

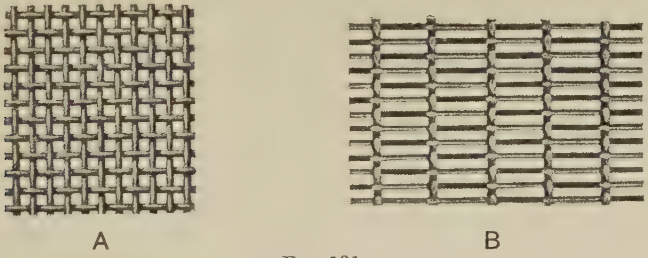

Fig. 131.

Woven-wire Screens.—A, double-crimped screen, normal square aperture. B, rolled screen with rectangular aperture (p. 220).

Commercially, woven screens are known by the number of holes per linear inch, this number being the screen mesh, or, more simply, the mesh. Though with the standard I.M.M. screens (p. 199), mesh has a precise relation to the actual aperture—being one-half the reciprocal of that aperture—the trade article does not maintain that relation. With each mesh three qualities of wire are used, these being known respectively as heavy, medium, and light, the first giving the smallest discharge area and the last the largest. Compared with these wire sizes, that used in the I.M.M. laboratory screens would be extra heavy.

Screens are also sometimes known by the trade number of the needle which will pass through the aperture. Each size of needle has a number; No. 3 needle, for instance, has a diameter of 0·0395 in., equal to 12 mesh I.M.M., and No. 7 a diameter of 0·0265 in., or 20 mesh. The limit to such a description for screens is, however, reached at about 30 mesh, since finer needles are not made. The ordinary range of woven screens, on the other hand, must be taken to include the 80 mesh, beyond which, as already stated, the 120-mesh and 200-mesh screens find employment in the laboratory.

Woven screens of silk thread are sometimes used in the sizing of fine dry ore. The fineness of such threads, their lightness and elasticity, permit finer screens to be made in this material than could possibly be made in metal wire. Such screens are flat, presenting their apertures in a manner favouring entry; they are light and can be stretched taut, a condition which reduces the chances of blinding; and even in the finest sizes they have a relatively large discharge-area. Silk screens as fine as 125—250 mesh have been used in the concentration of wolframite, for instance; they are expensive, and neither strong enough nor suited for general use.

SCREENING APPLIANCES

There are three principal types of screening appliance, namely, the Fixed Screen, the Shaking Screen, and the Revolving Screen.

Fixed Screens.—The simplest of all fixed screens is the fixed '*grizzly*,' which consists of a number of stiff steel bars laid on edge and at a set distance from one another, in a plane inclined downwards in the direction the ore is tipped from skip or truck (Figs. 9, 132). The set distance between the bars is generally 2 in. at the top of the bar section and more at the bottom, the bars being of wedge shape; common sectional dimensions are $\frac{3}{4}$ in. at the top, $\frac{1}{2}$ in. at the bottom, and 3 in. in depth. This greater aperture at the bottom ensures that material which enters at the top will have no difficulty in passing. These bars are 12—15 ft. in length. Through holes at the ends and the middle, three cross-bolts hold the bars together, the proper spacing being kept by washers. Usually so many are assembled that the total width of a single screen of this type is about 4 ft. 6 in.; where greater width is required several are placed together. For very large material old steel rails are often used, these rails being laid with the flange upwards.

The angle at which a grizzly is set is usually about 40°, though it varies from 25° to 50°, depending upon the nature and condition of the ore. If the ore be sticky the higher angle might be required, while the lower angle would be sufficient for dry lump-ore when the amount of oversize was not great. The object in placing the grizzly at an inclination is that it may clear itself, the oversize sliding and rolling to the bottom, leaving the undersize free to fall through. It is obvious, therefore, that if the material is delivered upon the appliance in the direction of the slope, the coarse oversize will receive a favourable impetus; the greater this impetus the less the necessary inclination. Again, if the proportion of oversize is small, the undersize will suffer little inconvenience if this oversize does not roll clear but awaits assistance from the hand; under such circumstance, also, the grizzly may be set at a low angle.

Bar screens are much used on collieries, though since coal may not be roughly handled the inclination of the screen may not be such as to add to any initial impetus. The screens used with coal are consequently set at a low angle and often quite horizontal. The assistance then necessary to secure progression of the oversize and fall of the undersize is generally obtained by moving the bars (Figs. 133, 134). It is usual to support alternate bars separately so that two sets of bars exist, each being connected to one of two eccentrics keyed at 180° to one another. When therefore one set of bars is rising and advancing the other is dropping and retiring. Each set in turn carries the material forward the length of the eccentric throw, and then drops back, passing its load on to the other, the progression of the material being assisted by the inclination of 10° or so. In the quiet rocking which accompanies this progression the undersize finds opportunity to clear itself and fall. This is the principle of the Briart and similar screens. It is not possible to apply such differential

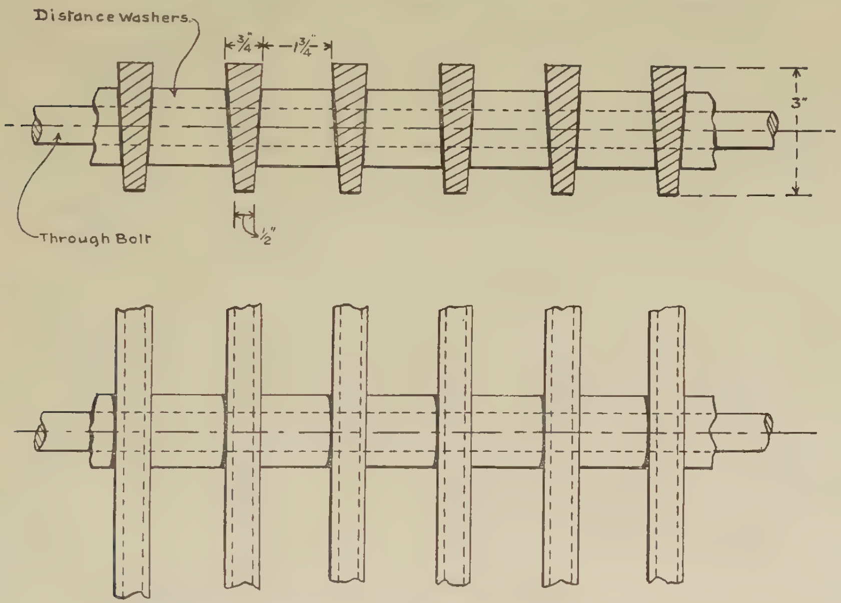

FIG. 132.

Grizzly.—Sectional Elevation and Plan. In part only (pp. 218-222).

movement of the bars to the screening of ore, because pieces caught between might break the bars.

In another type of bar screen used with coal, the longitudinal bars, being

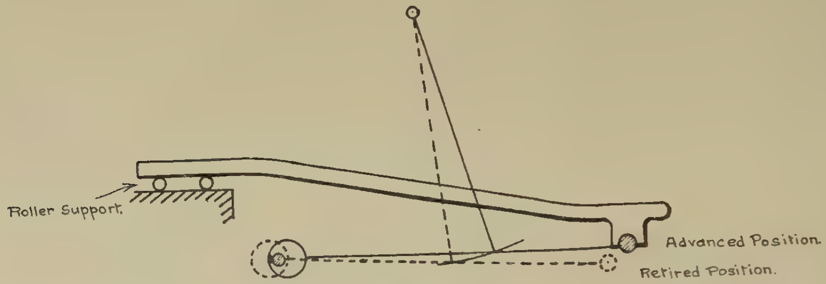

FIG. 133.

Moving-bar Screen.—Diagram of movement of one set of bars (p. 223). Alternate bars make a set. Of the two sets one is rising and advancing, the other falling and retiring.

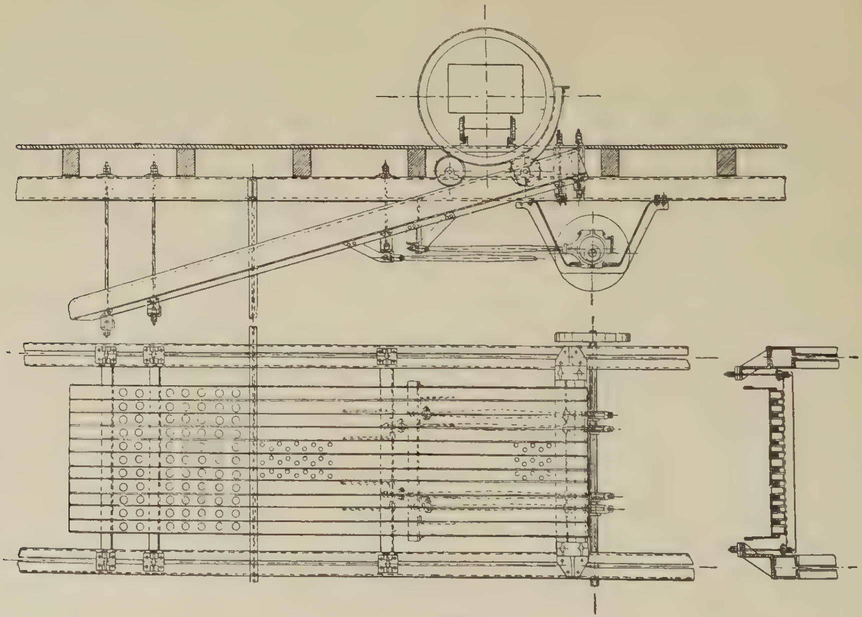

FIG. 134.

Moving-channel Screen.—Sections and Plan. In this screen, perforated channels take the place of bars. These channels, assembled in two sets of alternate bars, receive their motion from two eccentrics, one set being moved up while the other is drawn down. In the illustration only one eccentric is shown; the other, keyed at 180°, can be imagined. The position illustrated is that where the two sets are crossing, both then being in the same plane (pp. 223-225).

fixed, carry in proper recesses a set of transverse roller-bars, a more or less square aperture resulting (Fig. 135). Each roller-bar has a sprocket at one end, all these sprockets being rotated by a common chain in a direction to cause the desired progression. As a rule no inclination is necessary. Of such a design is the Borgmann screen.

In other designs the bars themselves are of channel iron with the base

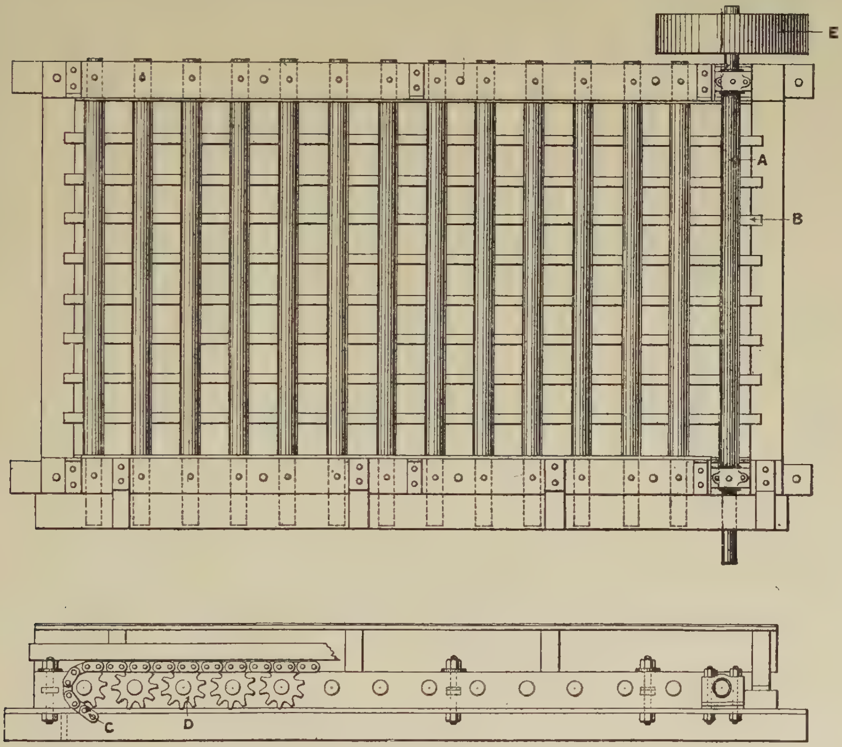

Fig. 135.

Roller-bar Screen.—Plan and Elevation. The longitudinal bars, B, are rectangular and laid on end. The transverse rollers, A, are round and laid in recesses cut in the others. At one and the same end of each roller is a sprocket, which is engaged by a chain common to them all, and by which they are all rotated in the same direction (p. 224) (Commans, *Proc. Inst. C.E.* Vol. CXVI.).

uppermost, while the sizing apertures are circles or squares cut in that base; the channels then have only just sufficient clearance between them to permit their movement in alternate sets (Fig. 134).

Fixed screens are also used for fine, dry material, though to no great extent. In the Rowand-Edison system of dry screening, the screens, inclined at 45°, in short steps with baffles between, are enclosed in a steeply-inclined or vertical chute, the material passing over ten or more screens in succession, each not more than 9 in. across. Fall being interrupted at each baffle, no acceleration of velocity takes place. Though fine ore when quite dry runs well and wears the screen but little, wear becomes considerable with moist damp ore.

Where perfect dryness is out of the question—and this is generally the case—screening of fine material must be undertaken under the conditions of the other extreme, namely, with the material borne in water. Fixed screens under such conditions are used with stamps, Chilian mills, Huntington mills, etc., not as independent appliances but forming part of the crushing machines, these machines in consequence being sometimes described as 'screen-faced machines.' With stamps the material is splashed upwards from the die against the screen, this being generally inclined and of woven wire because of the greater discharge-area such wire screens give. Square apertures are also generally preferred to rectangular because they give a more regular product. If rectangular apertures are used the longer dimension is disposed diagonally or vertically but not horizontally, since horizontal apertures present only their smaller dimension to the rising splash.

With Chilian mills and Huntington mills, the latter particularly on account of their high speed, rectangular apertures are preferred, the longer dimension being placed diagonally in the line of the pulp swirl. If placed otherwise the wear upon the screens would be considerably greater and the opportunity to discharge appreciably less.

Fixed screens, when these are independent appliances, grizzlies, for instance, generally demand the sacrifice of considerable height. This loss of height is mitigated, and sometimes even avoided, by the use of shaking or revolving screens.

Shaking Screens.—Ordinary shaking-screens or shakers are rectangular trays with bars or perforated plate for bottom. The tray, set at an inclination of about 10°, is fed with material at the upper end, which material it delivers at its open lower end. Assisted by the inclination, this progression from end to end is effected by giving the tray a series of shakes or bumps. If this shake be by cam action, the tray is usually so suspended that the cam forces it backward and upward, whence upon release it returns by gravity to strike against a stop, the screen being arrested while the material moves forward, a little each time (Fig. 136, A).

If an eccentric or crank be used the tray may be supported under the feed chute by wheels on a track, or it may be suspended by hangers. Then during the forward stroke the material goes with the screen, being prevented from returning during the backward stroke, partly by the inclination, partly by the pressure of newly-arrived material behind (Figs. 136, B; 138).

Sometimes, again, the design is varied by hanging the tray at its lower end and supporting the upper end directly on the eccentric, this direct attachment causing a considerable up-and-down movement additional to the reciprocation (Figs. 137, 140).

A special type of shaking screen is the Zimmer or Ferraris screen, the tray of which is supported upon rows of wooden laths or leaf springs having their common base upon girders, whereon also the shaft of a driving eccentric finds support (Fig. 136, C). These springs do not stand vertically, but lie over at an angle of about 65° towards the eccentric, which latter, by means of a connecting rod, pulls the tray down on its backward movement to let it fly forward and upward as the revolution becomes complete. In this last movement, which is assisted by the springs, the material leaves the tray, and, missing the next succeeding backward movement, returns only in time to receive another forward impulse. The progression thus becomes a series of hops, and is so positive that no assistance from inclination is needed, the screen commonly being horizontal.

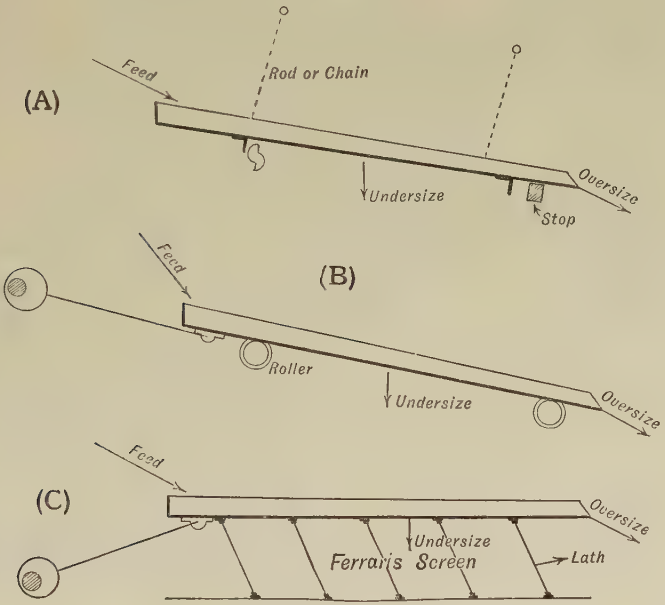

FIG. 136.

Shaking-screens or Shakers.—Diagrams of Types. A, cam-actuated tray swinging from rods. B, eccentric-operated tray swaying on rollers. C, eccentric-operated, spring-supported horizontal tray (Zimmer or Ferraris Screen) (p. 226).

Usually, for one simple separation of undersize and oversize, no great length is needed, and with ordinary apertures a tray 3 ft. wide and 6 ft. long would have a large capacity. For shaking screens in general, it may be said that with a 2-in. aperture the capacity would be about 50 tons of feed per square foot of screen area per day, this capacity becoming proportionally reduced as the aperture decreased, till at one millimetre it would be about one ton per square foot.

Where, however, it is desired to make several classes, the necessary divisions may be accomplished either by making the tray longer to receive other screens of increasing aperture each in its turn separating an undersize, or by making it deeper to accommodate screens of decreasing aperture each in turn separating an oversize (Figs. 139, 140). With 'compound undersize' screens the large oversize is the last product to be separated, with 'compound oversize' screens it is the first. The long tray of the former with the several screens all in one plane has the advantage that these are always open for inspection and can readily be removed. Sometimes, to obtain similar conveniences with the short but deep nest of the compound undersize screen, the undersize from the first

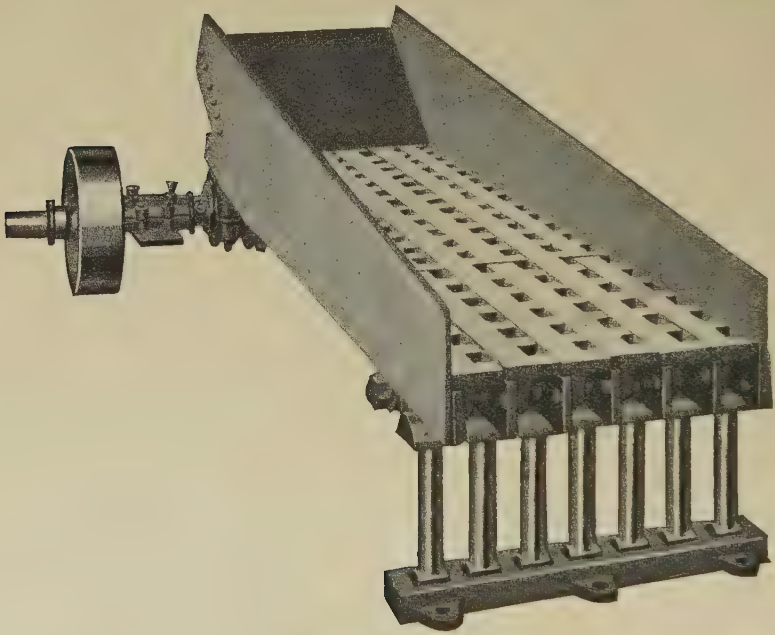

FIG. 137.

Shaker for Coarse Material.—General View. Heavy tray directly supported upon eccentric strap at the upper end and upon standards at the lower end (p. 227).

screen is directed by a dead plate to the head of the next screen of finer aperture inclined in the opposite direction, and so on; or the several screens may be separate and with sufficient space between them for convenient manipulation.

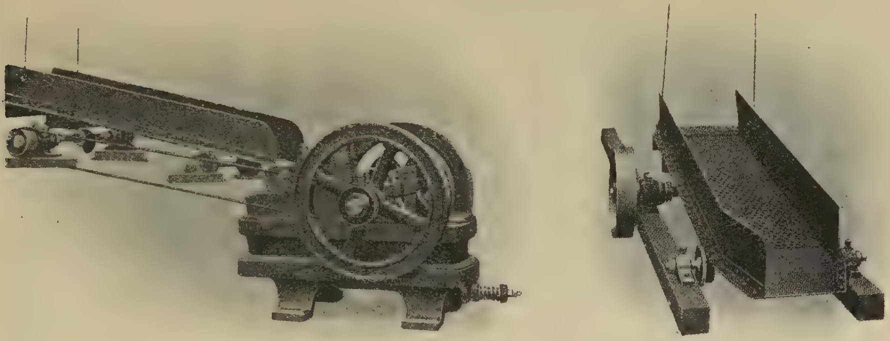

FIG. 138.

Shaking-screen used as Feeder to a Breaker.—General View. Tray cam-operated; back end swung from rods, lower end on rollers (p. 227).

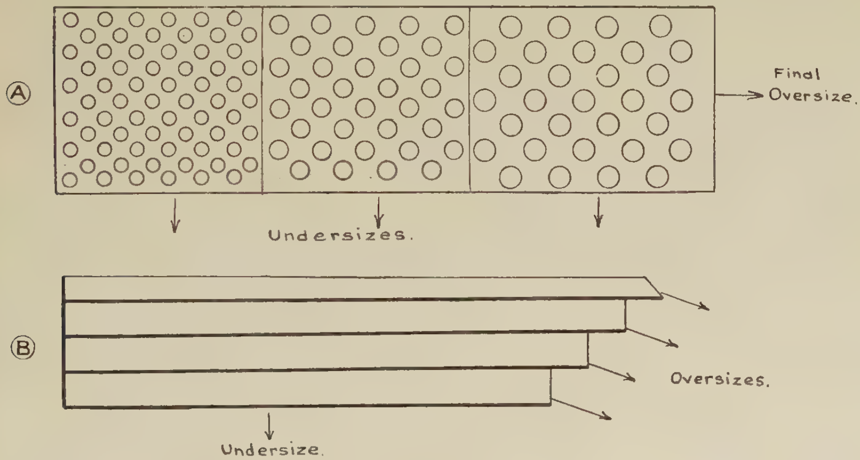

FIG. 139.

Compound Shaking-screens.—Diagrams. A, plan of an oversize screen. B, sectional elevation of an undersize screen (p. 228).

In all these devices the speed and amplitude of the shake will largely depend upon the size of the material; when this is large the speed will be less but the amplitude greater, while when it is small the reverse will obtain. Ordinarily the number of complete shakes varies from 80 to 300 per minute and the amplitude from $1\frac{1}{2}$ in. to 8 in. Where the number of reciprocations is great and the tray heavy, much vibration is caused unless the movement is balanced either by special weights or by running two screens together, the inward movement of one balancing the outward movement of the other (Fig. 141).

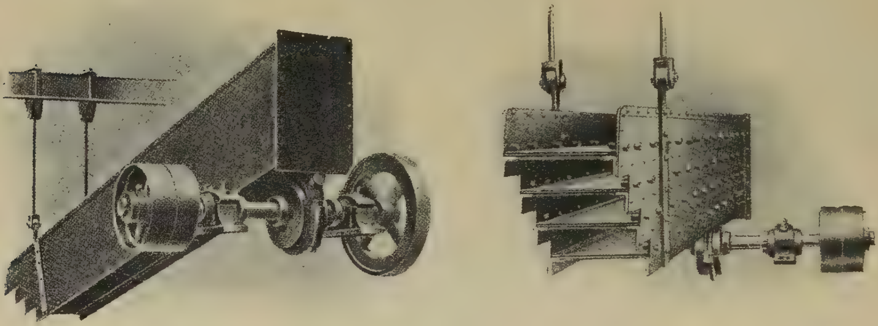

FIG. 140.

Compound Oversize Screen.—General Views. Nested tray swung from hangers at the lower end and supported by rigid connection to the strap of an eccentric at the upper end ; shake imparted by the same eccentric (pp. 227, 228).

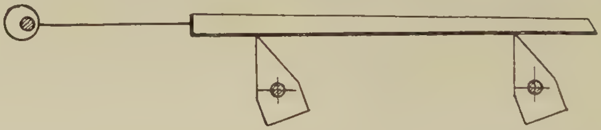

FIG. 141.

Balanced Shaker.—The rocking supports are weighted below their axes (p. 230).

Gyratory Screens.—Closely related to these shaking screens with reciprocating motion are others having a gyratory motion, whereby continual reversal of direction is avoided and smooth running secured. Such a motion though agitating the material most efficiently, effects no progression, and there is consequently no advantage in making the screens long ; such gyratory screens accordingly are square in shape and generally about 4 ft. in dimension. The Coxe screen of this type consists of a nest of such screens with the coarsest on top ; these screens, not being quite plane but somewhat dished, have their lowest point at the front, where discharge of the oversize takes place. In operation, this nest is gyrated in a horizontal circle upon four double-cone rollers, one at each corner, by means of a crank having a throw of about 2 in. and making about 150 r.p.m., this crank being at the upper end of a vertical driving spindle underneath (Fig. 142). In another design, the Karlik screen, a similar movement is conveyed to a nest of screens suspended from a high central support.

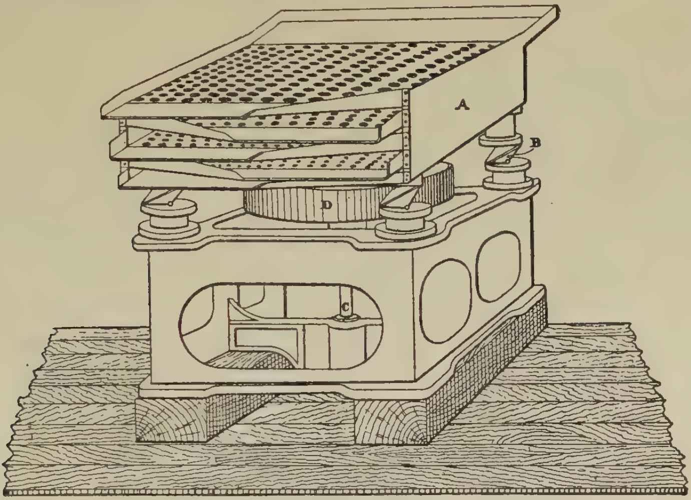

FIG. 142.

Gyratory Screen (Coxe).—General View. Screen driven by a crank of small throw engaging the bottom of the nest. A is the nested tray; B, the double-cone roller; D, a heavy disc to balance the movement (p. 230) (Commans, *Proc. Inst. C.E.* Vol. CXVI.).

Gyratory screens, lying more or less horizontally, collect a bed of material which becomes stratified, the larger pieces riding on the smaller, a distribution which facilitates passage of the undersize and discharge of the oversize. They have been used largely to divide anthracite into trade sizes, but also in the dressing of manganese ores, etc.

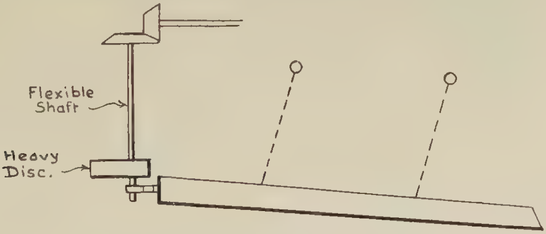

FIG. 143.

Vibromotor Screen.—Diagram (p. 231).

Vibromotor Screens.—A somewhat similar motion is given to the class of screening machines known as vibromotor screens. With these a rectangular tray hung by flexible hangers at an inclination sufficient under agitation to cause progression and eventual discharge, is attached to the lower end of a flexible shaft capable of being revolved at a high speed from above, and having near its lower end a heavy metal disc disposed eccentrically. During revolution the lower end of the flexible shaft describes a roughly circular path, a motion which, somewhat modified by the manner of suspension, is conveyed to the screen (Fig. 143).

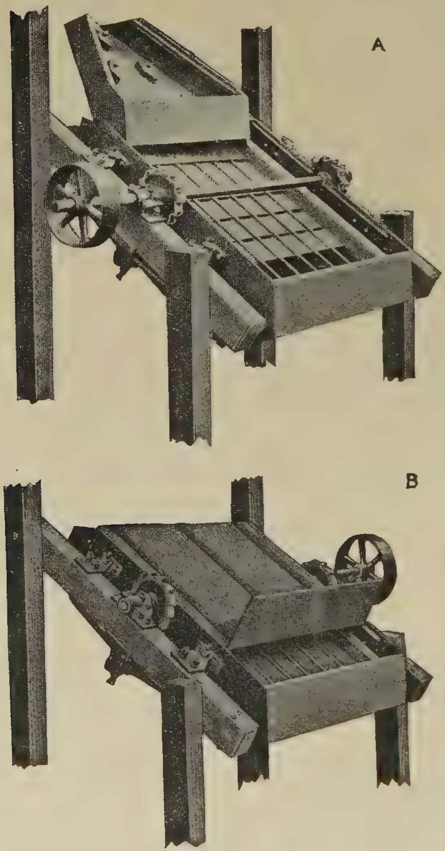

Fig. 144.

Impact Screen.—Spring-operated Type. A, open screen for fine material in water suspension. B, closed screen for fine dry material (p. 232).

Impact Screens.—Last among these shaking screens comes a type where the movement, instead of being in the plane of the screen, is at right angles to that plane, the effect being as though the screen were tapped with a hammer. Such screens are known as impact screens. The double effect of this shake is to re-shuffle the material, and to keep the apertures free. To obtain the necessary progression from feed to discharge, these screens are set at an inclination of about 40°. The blow may be given in this wise: by every tooth of a ratchet wheel or every arm of a cam, one such wheel or cam being on each side of the frame, the screen frame is pressed down upon leaf or elliptical springs, which at the passage of tooth or arm force the frame smartly up against stops, producing thereby the knock which characterizes the machine, the number of such knocks being generally about 600 per minute. The ordinary size of such a screen is 3 ft. by 4 ft. and the screens themselves are generally of woven wire;

when used to screen fine material of say 60 mesh, the capacity is about 4 tons of feed per square foot of screening surface per day; with larger aperture—and occasionally an aperture of 1 in. is used—the capacity is proportionally greater. Impact screens may be used wet or dry (Fig. 144).

Others of these impact screens actually use small hammers spaced regularly over the surface, and operated in succession mechanically. With these the screen is automatically held taut, and specially backed at the points where the hammers tap. This type of machine is designed for dry screening; it has been used with apertures varying from $\frac{1}{4}$ in. to 120 mesh (Fig. 145).

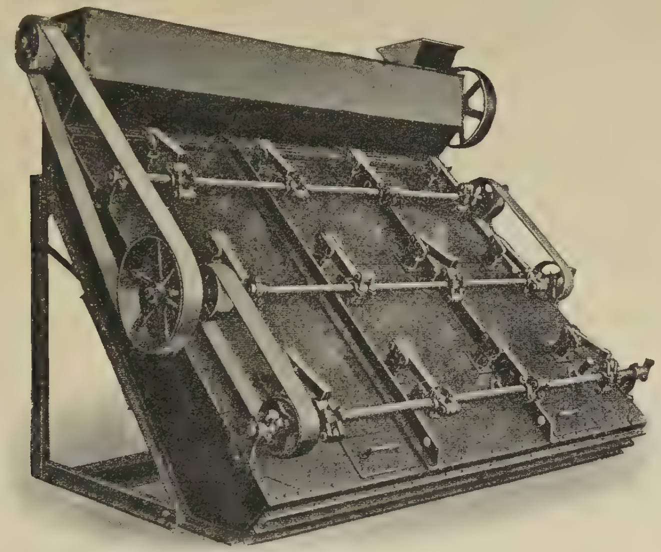

Fig. 145.

Impact Screen.—Hammer-operated (Newago) Screen. Closed screen for fine dry material (p. 233).

In recent years a screen beaten by the unbalanced running of a specially-designed motor beneath, has been in use upon the large copper mines in Arizona, under the name of the Mitchell screen (Fig. 146).

When screening very-fine dry material upon a shaking machine with gyratory motion, the beating of the sieve may be done by small, hard bodies placed upon the screen itself. Pieces of hard rubber or small metal discs so placed will minimize blinding and render screening more effective. Gryatory screens being practically horizontal, the beaters may be kept regularly distributed by a special wire frame laid on them, each beater being limited to one square of that frame.

Shaking-screens of whatever type have the advantage that practically the full area of the screen is actively engaged and a minimum screen-area suffices. Ordinary shaking-screens have the further advantage that concomitantly with the actual screening there is a progression of the oversize, and the screening appliance serves to that extent as a conveyor. Requiring little height of themselves they are often conveniently suspended as feeders to breakers or other crushing machines (Fig. 138). Should, however, the material be clayey, the agitation, even with generous use of water, is not sufficient to release the individual pieces, and efficient sizing becomes impossible. To increase the agitation by increasing the number

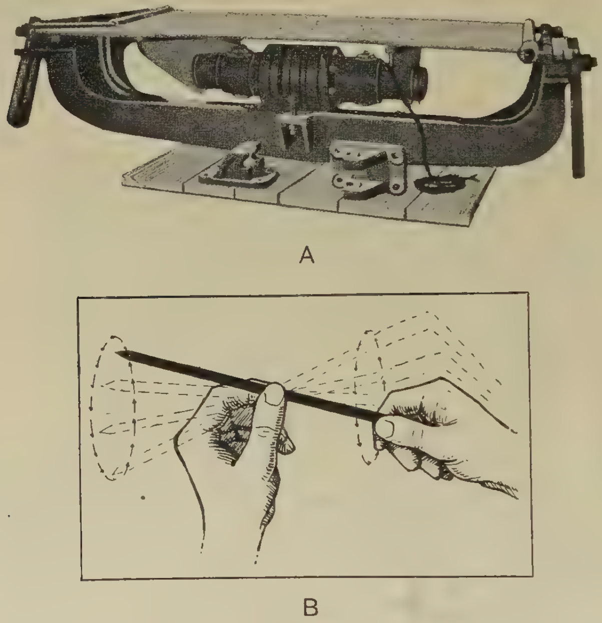

Fig. 146.

Impact Screen.—Motor-operated (Mitchell). A, general view. Under the stretched wire-screen is a tube-shaped motor making about 3600 r.p.m. The relatively long shaft of this motor has at either end a cage containing steel balls free to fly out. At one end these balls are confined to one-half the circumference, and at the other end to the opposite half, with the result that the shaft does not run truly but wobbles, and in wobbling delivers a blow which is conveyed to the screen by arms at either end. This blow is an upward rotary movement of about $\frac{1}{8}$-inch amplitude. The screen is suitable for coarse or fine and for wet or dry material. Its standard size is 4 ft. by 6 ft. (p. 233). B, diagram of movement.

and amplitude of the shakes would incur great increase in power consumption and machinery repair. For such clayey material and wherever a moderate amount of washing is desired, revolving screens are more suitable.

Revolving Screens.—The most important appliance of this class is the Trommel, the screen plates of which, instead of being plane surfaces, are bent around a cylindrical or conical framework, which in turn is supported either by spider arms from a central shaft, or by encircling tyres running upon rollers (Figs. 147, 153). The cylindrical trommel is laid with its axis at a slight inclination, so that upon rotation the material fed into it at the upper end progresses towards the lower, with continuous passage of the undersize through the walls, the oversize remaining for discharge at the end. The path taken in this progression through the trommel is obviously not a simple straight line. In the rotation the material is caught and carried up the rising side till, gravity overcoming centrifugal force and friction combined, it leaves the circumference and slides downward in the direction of steepest slope, to be finally arrested by friction, rise then beginning again (Fig. 148). The course taken is therefore zigzag, an upward move in the plane of rotation followed by a downward move under gravity and the pressure from oncoming material. Both movements having a forward component it needs but their repetition to ensure eventual discharge, the number of the repeated movements, all other things being equal, being within limits a fair measure of the completeness of the screening effected. These considerations, though evolved from the behaviour of a single piece, apply equally to the mass within a trommel.

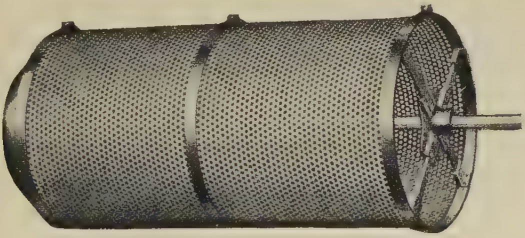

FIG. 147.

Cylindrical Trommel.—General Appearance. The spider arms supporting the rings are well shown, as also are the outside straps by means of which the screen-plate is held around those rings. At the feed end a short conical shield contracts the entrance, preventing backward spill of material (p. 235).

With regard to speed of rotation, there is a critical speed at which centrifugal force would hold the oversize at the circumference for the complete revolution and make progress impossible. This condition, as was shown when discussing tube-mill speed, would be reached when $n = 76/\sqrt{d}$. It is evident, therefore, that speed has a good deal to do with the height to which the material rises before it begins to slide back. In practice the speed is generally such that the tangent to the trommel circumference at the point to which the material rises, makes an angle of 40°, more or less, with the horizontal, experience having shown this to give the best results (Fig. 148). With the diameters commonly employed, this speed is generally 10—20 r.p.m., and less than half the critical speed.

In general, the extent of the backward slide brings the material little beyond the vertical plane passing through the trommel axis. Between the extreme positions of rise and fall lies the segment of the screen circle occupied by the material, a segment generally not greater than 30° to 40°—or about 10 per cent of the total screening - area—this giving under ordinary conditions of diameter and inclination a bed of about 2 in. of material. To occupy a greater segment would mean such a depth of bed that satisfactory screening would be endangered. The amount of material undergoing sizing within the trommel will accordingly depend upon the trommel diameter, a large diameter having therefore larger capacity. In practice the diameter is generally from 2 to 5 ft.

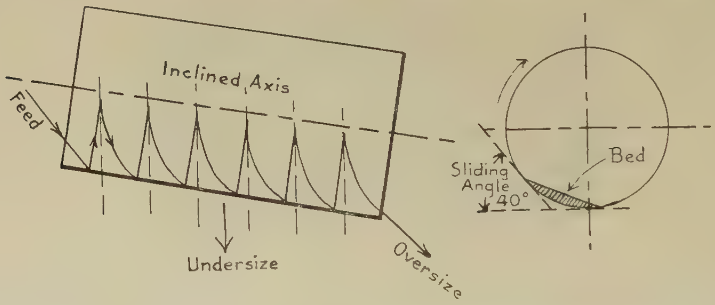

FIG. 148.

Cylindrical Trommel.—Diagram of Movement (p. 235).

In addition to its influence upon the height to which the material rises, speed also influences the rate at which the oversize is discharged, since it determines the rate at which rise and fall are repeated. Of greater influence, however, is the inclination, this largely determining the progress achieved each revolution. When the inclination is great this progress is great also, and the zigzag path of the material is shortened. The limit of inclination is reached when owing to the speed of passage the undersize has no adequate opportunity to drop out. Ordinarily this inclination varies from 2° to 15°, an inclination of about 5° being common.

With respect to length, all other conditions being equal, the longer the trommel the greater the number of repeated movements and the more perfect the screening. Obviously, any length beyond that necessary to secure satisfactory sizing is a wasted provision. Ordinarily, the length of a single trommel making one separation of undersize and oversize varies from 5 to 15 ft., long length being associated as a rule with finer material, which is more difficult to size. Where, however, the trommel is applied to clean as well as to size—as for instance on dredges working alluvial deposits—it may well be longer and of greater diameter than the dimensions indicated, trommels of 6 ft. diameter and 20 ft. in length being not uncommon for such work (Fig. 149).

Reviewing the foregoing considerations, it is seen that the trommel, like the shaker, in process of fulfilling its function as a sizing machine effects also the regular progression of the oversize from feed to discharge. Its capacity, measured in terms of original material efficiently screened, increases proportionally with the diameter. Up to a certain point it also increases proportionally with speed and inclination. To length, however, capacity is not entirely proportional, because if the feed be at such a rate that the bed at the upper end is not too deep, the diminished bed at the lower end will not fully occupy the screen. Much, of course, depends upon the material; that which contains a large proportion of pieces of size similar to the aperture, is difficult to size. Fine material, for instance, or coarse material with small range of size, presents this difficulty; with such material the necessarily large number of repeated movements is obtained partly by low inclination and partly by length of trommel. Where, however, the proportion of undersize is large, screening is relatively easily effected. In general, and assuming about 50 per cent of oversize, a trommel having apertures of 1 mm. should receive and screen

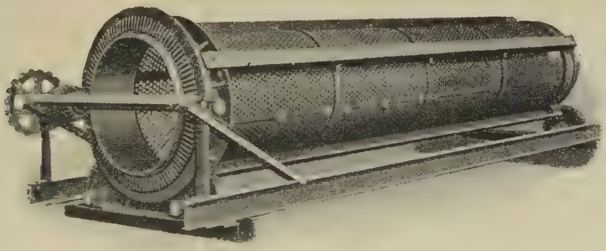

Fig. 149.

Heavy Cylindrical Trommel.—Roller-borne and driven from discharge end. Rings attached to outside bars which again are secured to cast-iron heads, machined to have steel tires shrunk over them. The rings mark separate lengths of screen plate, this plate being of ordinary steel punched, or of manganese-steel cast. Screen carried on chilled-iron rollers (p. 237).

half a ton per square foot of screen area per day of 24 hours, the capacity with greater aperture increasing in direct proportion to the aperture.

The separation of the undersize and the progression of the oversize, which with the cylindrical trommel were obtained by rotation about an inclined axis, are equally well obtained by rotating a conical trommel about a horizontal axis (Fig. 150). The material, being then lifted in the vertical plane of rotation, moves towards discharge only during fall, this fall, in consequence of expanding diameter and of pressure from oncoming material, being directed forwards. Such conical trommels have the disadvantage that the undiminished feed is delivered at the small end, which must be overloaded if the far end is to be loaded effectively; they, also, are not so easily repaired. On the other hand, the horizontal axis presents compensating conveniences in arrangement and driving mechanism.

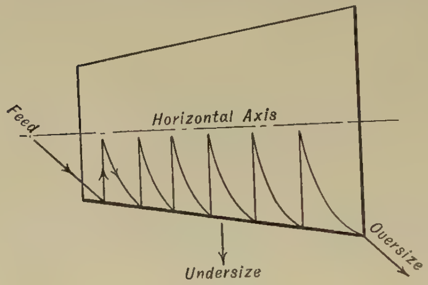

FIG. 150.

Conical Trommel.—Diagram of Movement (p. 238).

Another design of trommel is that with a uniform polygonal section. Such a section allows the screening surfaces to be independent plates, separately replaceable when broken, a substantial advantage with woven screens as these are liable to break unexpectedly. It is further claimed that this type of trommel, which is generally hexagonal in section, causes greater agitation of the bed, and consequently gives, with fine dry material, greater efficiency; it is, however, not much used.

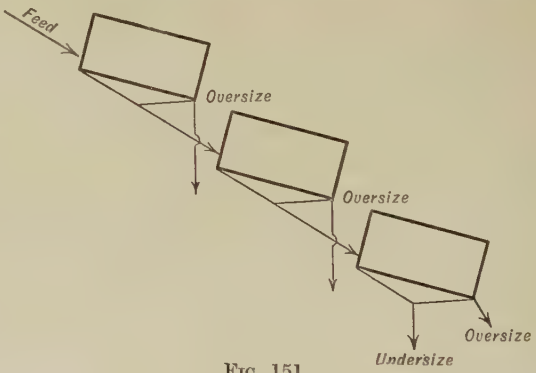

FIG. 151.

Trommel Series.—Diagram. Products consist of several oversizes and one undersize (p. 238).

The two products made by a trommel having but one size of aperture are often insufficient, a greater subdivision may be necessary. Such multiple division may be obtained by placing two or more trommels in series, of which the uppermost having the largest aperture separates an oversize, passing its undersize as feed to the next succeeding machine, which in its turn separates an oversize, and so on (Fig. 151). This arrangement of an 'oversize series,' while permitting the trommels to be simple and small, is relatively complicated in its driving mechanism and necessitates some height; where, however, the last screen is fine and consequently weak, such a series possesses the considerable advantage that only fine material reaches that screen.

But, where even the finest screen is coarse and strong, instead of a series, a 'compound cylindrical trommel' with apertures of increasing diameter from entry to discharge may be employed; the finest screen, coming first, then bears the weight and wear of all the material (Figs. 152, 153). Such a trommel, avoiding the multiplicity of parts which a series demands, offers the advantages of simplicity in driving and saving in height. From each screen the undersize is the finished product, the oversize passing on to the next screen for the separation of a further undersize, and so on till one final oversize remains; these trommels, accordingly, may be described as 'compound undersize trommels.' Three separate screens are as many as can conveniently thus be compounded in one machine. Where the amount of oversize is great the finest screen is sometimes shielded by a coarser screen, though this arrangement detracts from the simplicity of the appliance; where also screening is undertaken dry, that finest screen is commonly enveloped in an outer hood to prevent the diffusion of dust (Fig. 153). It is not surprising therefore that compound cylindrical trommels are generally both long and heavy, necessitating support upon rollers and driving from the periphery.

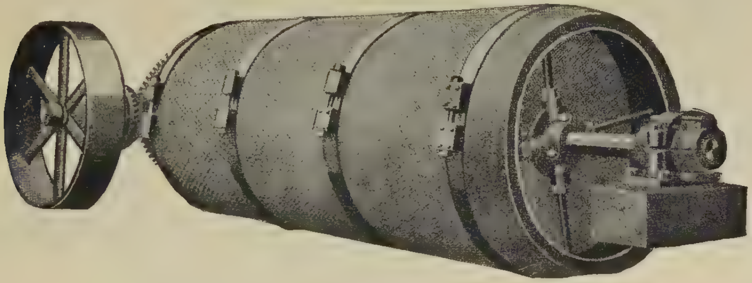

FIG. 152.

Compound Cylindrical Trommel (shaft-supported).—Driven by bevel pinion engaging the periphery at the discharge end. The attachment of the different screen plates by straps over interior rings is well illustrated, as also are the spider arms (p. 239).

Finally, comes the arrangement of several concentric conical screens, the coarser aperture innermost and the finer outermost, there being generally not more than four in all (Fig. 154). With the feed entered at the centre on to the coarsest screen, an oversize becomes the finished product, the undersize passing to the enveloping screens each of which in turn separates a further oversize; these trommels, accordingly, are 'compound oversize trommels.' It will be noted that the inner screen of smallest extent must accommodate the undiminished feed, of which the outer screen of greatest extent receives but a fraction. Under such conditions and to avoid the difficulties of its untimely renewal, the inner screen must be particularly strong. At best, however, these conical compound trommels are complicated in construction. They are not largely employed in ore-dressing.

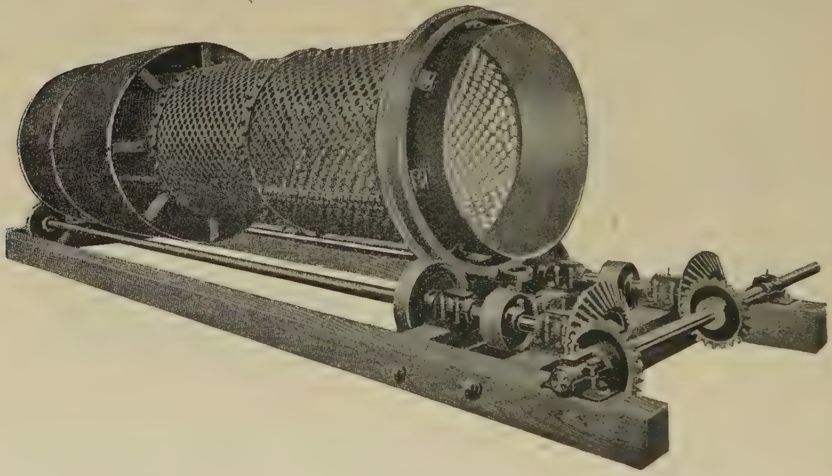

Fig. 153.

Compound Cylindrical Trommel (roller-supported).—The screen plates of this trommel are of sufficient strength to constitute—when riveted to one another—a shell which needs neither support from an interior axis nor from external bars. This trommel is driven entirely by rollers, through tires at either end. Downward thrust due to inclination is taken up against a roller stop, clearly indicated. At the finer end is a dust hood or jacket (pp. 235, 239).

The range in size of material ordinarily screened by trommels is from 4-in. ring to about $\frac{1}{24}$ in., though material still finer is occasionally handled. Concerning the screens, punched screens are generally used down to about $\frac{1}{8}$ in., below which woven screens are better. Punched holes though often round are best slotted, the length of the slot being placed in the plane of rotation, so that when foreshortened as the trommel revolves the effective aperture suffers little reduction. The plate is generally of ordinary steel; when of manganese steel it may be thinner, blinding less. The life of a manganese-steel screen is generally measured in years, while that of a screen of ordinary steel is more often measured in months.

Some trommels are worked wet, others dry. Water, when used, is generally played on to the rising side of the screen from a longitudinal row of jets outside, or it may issue from a similar row inside. Where the material is relatively fine, the entry of water from the outside assists the oversize to fall back, clearing the screen and decreasing the chances of blinding. Where the feed is water-borne the accompanying water leaves the solid particles piled up at the entry, and the continual play of additional water becomes necessary for the progression of the oversize and the passage of the undersize. Water also assists in bearing fine products away, and without it the discharge chutes must be given greater inclination. It increases the wear, but lessens the noise and subdues the dust. With large material these advantages of water are either not obtained or are of little moment, and the screening of such material often is accomplished entirely dry.

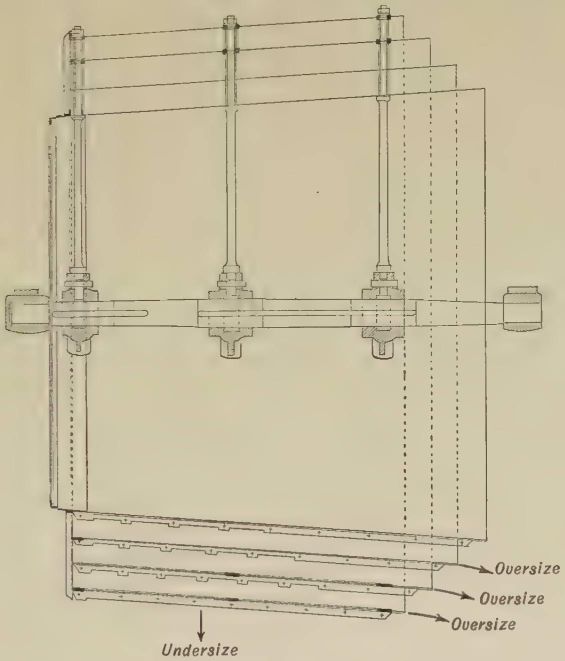

FIG. 154.

Compound Conical Trommel.—Sectional elevation. Built on spider arms. Oversize separated by each screen, the outside screen separating in addition the final undersize (p. 240).

It will be realized from the foregoing that trommels, generally speaking, are coarse-screening appliances. Doing such work they find their greatest application in association with coarse-concentrating appliances, and principally with jigs (Fig. 222). Another important service they perform is that of cleaning the ore preparatory to picking, to crushing, or to their own proper function of sizing (p. 20). The large amount of movement not only of the screen itself but of each particle, permits them to do this cleaning completely. For the sizing of fine material, they are, however, inefficient, largely because most of the material is then of much the same size as the aperture, but also because the small weight of the undersize particle is insufficient to force a passage; finally also, because if water be added to assist, the trommel movement is no longer the prime factor in the operation and the trommel itself loses much of its justification.

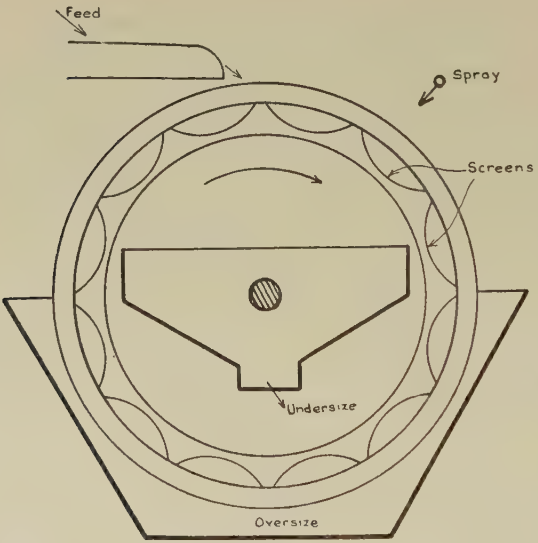

Fig. 155.

Drum Screen (King screen).—Diagram (p. 243).

Wet-screening Machines.—Dry fine-sizing, as has already been described, can be accomplished by passing the material over fixed or impact screens set at a high angle, or by shaking it upon flat screens. Since, however, most fine-crushing is undertaken in water, wet fine-sizing is more appropriate. This wet sizing is largely accomplished by water in appliances known as 'classifiers'; but, as will be described later, the water-sizing thereby effected permits fine particles of mineral to associate themselves with coarser particles of gangue, a condition of things not always desirable; moreover, much additional water is required. Often, therefore, wet sizing by screening offers advantages which more than outweigh the simplicity and reliability of water-sizing.

Wet-screening machines for fine material all follow the idea that the stretch of screen on to which the stream of pulp falls must continually move forward, to carry the oversize away and to permit a clean stretch to move up into place.

This forward movement may be accomplished, for instance, by placing a revolving screen to receive the falling stream much as the overshot water-wheel receives its water. Of this design, the King screen developed at Broken Hill, N.S.W., is a circular drum about 5 ft. in diameter and 3 ft. in width, around which woven-wire cloth is wrapped and fixed, sagging between twelve regularly spaced, radial supports (Fig. 155). Upon this screen the pulp stream falls from a distributing lip, the undersize passing through into a centrally-disposed collecting pan, the oversize passing on till, as the screen turns over, it falls, its detachment being assisted by water jets. Of the undersize, the greater portion passes through at once, but before the oversize is detached it passes under a row of water jets to be further cleaned, the additional undersize thus separated also falling into the collecting pan. The oversize is caught in the external housing.

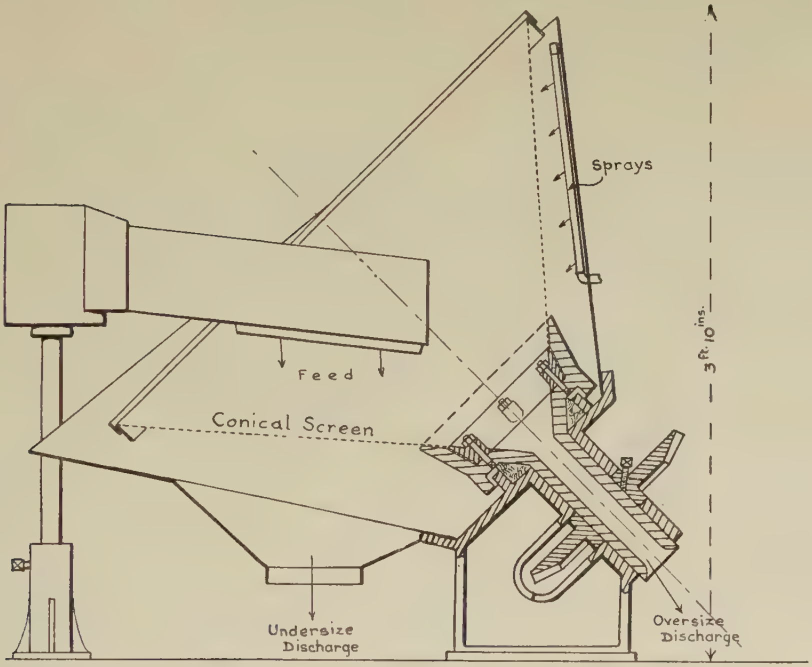

FIG. 156.

Cone Screen (Bunker Hill screen).—Sectional diagram (p. 244).

A very similar "drum-screen" is the Ford screen developed at Joplin, Missouri, in which ten flat screen-bottomed trays complete the circle. Both this and the King screen are simple, reliable, and cheap in operation. They have been used with satisfaction upon material large enough for an ordinary shaker, and upon material as fine as 30 mesh.

A screen of different type, but similar in that the geometric axis of the machine is the axis of rotation, is the "conical screen," typically represented by the Bunker Hill screen (Figs. 156, 157). In this the wire cloth, instead of being wrapped around a drum, is the stiff lining to a conical framework set apex downward with its axis at an inclination of 45°. Into the open base of this cone the pulp is fed, the undersize passing through while the oversize is carried upwards till the flat generatrix upon which it fell at entry has become so steep that, with the assistance of water played from the outside, the oversize falls to the apex, where, through the hollow spindle by which the conical frame is rotated, it makes its exit. This simple and effective screen has been used for material as fine as 30 mesh.

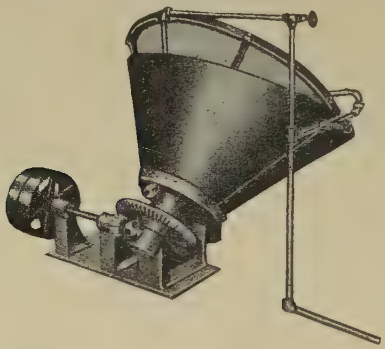

Fig. 157.

Cone Screen (Bunker Hill screen).—General view (p. 244).

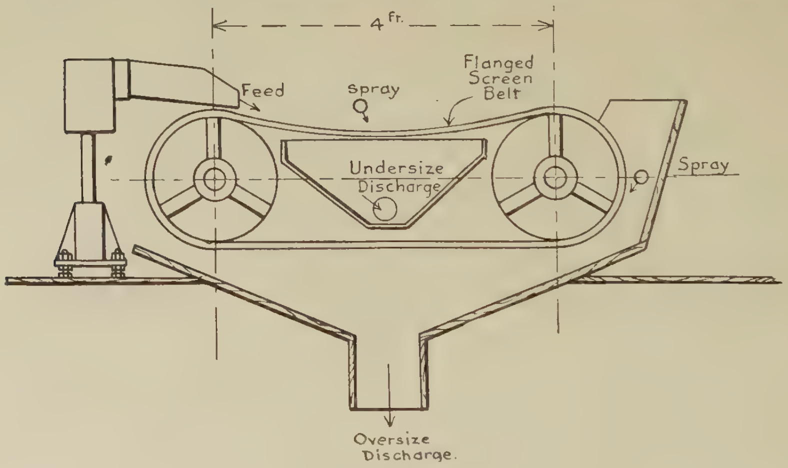

Fig. 158.

Belt Screen (Callow screen).—Diagram (p. 244).

Applying the same principles but using an endless woven-wire belt running round two rollers, the "belt screen," represented by the Callow screen, is an efficient sizing-machine for fine material (Fig. 158). Ordinarily, this belt is 2 ft. wide and the rollers, spaced with centres 4 ft. apart, are 12 in. in diameter. On to this belt, which can be moved forward at a speed varying from 25 to 125 ft. per minute, the feed pours from a distributing lip, the undersize passing through into a collecting pan, the oversize, after passing under one or more rows of water jets to be cleaned, being finally detached with the assistance of water as the belt turns round the forward roller. The cleaning of the oversize being possible over the greater portion of the horizontal forward traverse of the belt, these screens are efficient at normal speeds of running, even with material so fine as 60 or 80 mesh. Owing, however, to continual bending as the belt passes round the rollers, the wire cloth has but a short life and renewals are frequent. The machine is constructed to facilitate this renewal, in addition to which two machines are considered to make a unit, of which, when one is under repair, the other is speeded-up to take the whole feed (Fig. 159). The cost of belts with these machines is considerable, this cost being increased by the necessity to provide such flanged edges as will secure their smooth running against the roller flanges.

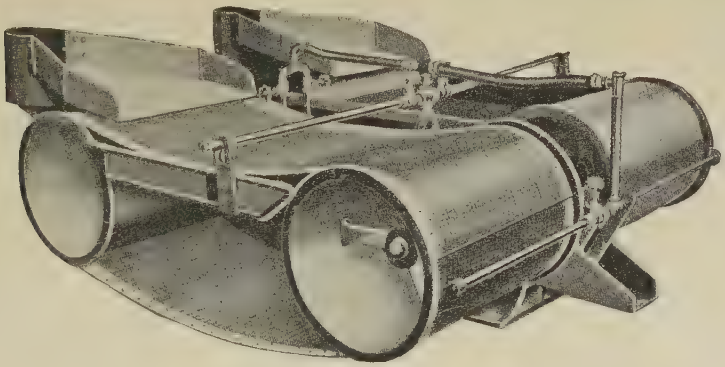

Fig. 159.

Belt Screen (Callow).—General view of Duplex type. In this duplex arrangement of two belts, one is capable of being speeded-up to take the whole stream at such time as the other is under repair or being renewed. The driving arrangements, by means of stepped pulleys, include provision for an easy change of speed. The two undersizes flow together and are discharged centrally (p. 244).

The capacity of these wet-screening machines measured in terms of screening area is greater than that of the ordinary revolving-trommels. At an average speed, for instance, the capacity with an aperture of one millimetre is about 4 tons per square foot of screening surface per 24 hours, which is about four times that of a shaker and eight times that of a trommel of the same aperture. For smaller apertures the capacity would be proportionately smaller.

On the other hand, wet screening is not so perfect as dry screening, that is to say, it is not so close to the aperture of the screen. A 30-mesh screen when worked wet would, for instance, deliver such an undersize as might be expected from 40 mesh worked dry. This disparity arises from the fact that in wet screening the undersize particles have generally only two chances to get through, one with the falling stream and the other when the oversize is cleaned under water jets, whereas in dry screening a further chance is offered with each shake of the screen or each revolution of the trommel, and the difficult particle, that is, one close in size to the aperture, at last passes.

The pulp fed to wet-screening machines has generally a dilution of about four parts of water to one of ore, while the additional water necessary for their operation varies from about half a ton to one ton per ton of ore. This low amount of additional water is one of their advantages; it will be seen later that water-sizing appliances demand much more.

Screening Efficiency

The efficiency of a screening appliance is measured by the ratio between the weight of obtained undersize and the weight of undersize determined by sizing analysis to exist; it is usually expressed as a percentage. By making sizing analyses of the feed and of the oversize —using a sieve having the same aperture as the machine—it is not actually necessary to trouble about these weights, the efficiency can be calculated without them. Thus: assuming, for example, the feed to consist of 25 per cent oversize and 75 per cent undersize, and the resultant oversize to contain 25 per cent undersize; then from 100 tons of feed the 25 tons of real oversize would be accompanied by 8·3 tons of undersize; and of the original 75 tons of undersize, 66·7 tons or 88·8 per cent would have been separated. The sizing efficiency in this case would therefore be 88·8 per cent (p. 293).

Given the design, disposition, and speed of a sizing machine, efficiency will largely depend upon the proportion of undersize in the material; as this proportion increases so does the efficiency. It will also depend upon the range in size of the particles present; where this range is small a division by sizing will be both difficult and inefficient. Again, efficiency depends on whether the operation is conducted dry or wet; if conducted wet the undersize has but one or two chances to pass, and a greater proportion of it will be discharged with the oversize. Meticulous closeness to the aperture of the sizing appliance is of course not everything; when screening wet the desired undersize would be realized by using a slightly coarser screen. In ordinary practice efficiencies of 80—90 per cent are common, while an efficiency of 95 per cent is not unusual with coarse material.

With increasing fineness both efficiency and capacity diminish till eventually it becomes better and cheaper to replace screen sizing by water-sizing (Fig. 160).

Finally, though screening is generally a preparatory operation, it is sometimes of service in giving final products, either of discard or enrichment. Notably, for instance, a final discard by screening appliances is commonly made in the beneficiation of alluvial deposits of gold and tin, the greater portion of the material being separated as impoverished oversize, leaving only the smaller portion for further treatment; in working stanniferous deposits by dredges it is the practice to discard at once somewhat more than half the material as the oversize from a trommel with $\frac{3}{8}$ in. holes (p. 348). In ordinary milling and dressing it is likewise not uncommon that a stage is reached where by screening an enriched undersize is obtained (Fig. 161). Contrariwise, when treating some surficial manganese deposits where clean manganese minerals occur as hard pieces associated with clayey material, screening may produce an

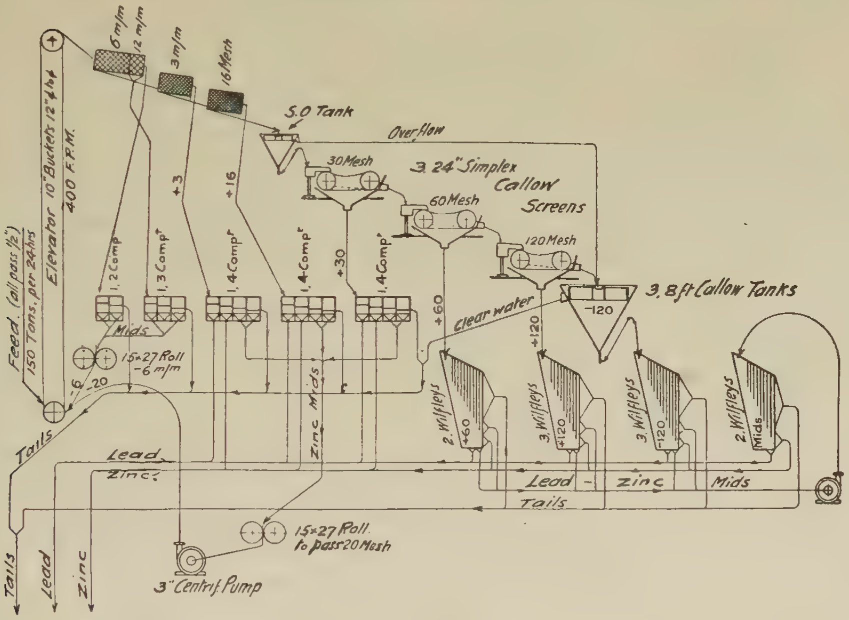

FIG. 160.

Sizing System at the Success Co. (Wallace, Idaho).—Ore, a heavy zincblende with galena and pyrite in a siliceous gangue. Trommels separating materials for jigs; Callow screens preparing feed for fine jigs and sand tables; water-sizing classifiers taking care of the finest material (p. 247).

oversize rich enough for the market, while discarding an impoverished undersize.

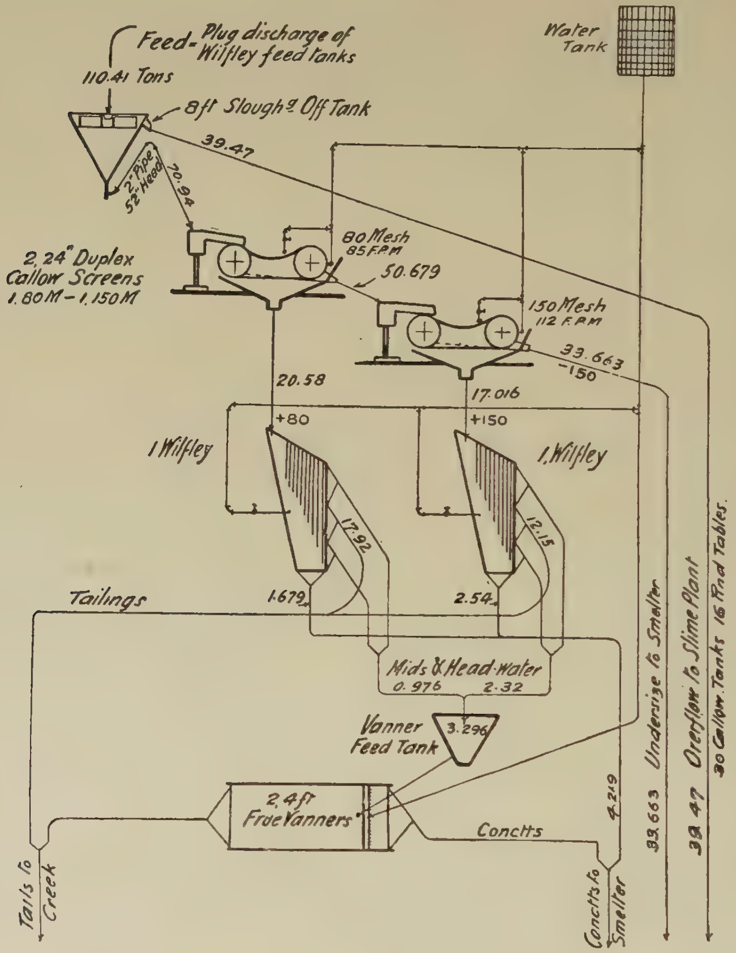

Fig. 161.

Fine Sizing at the Boston Montana Co. (Great Falls, Montana).—Water-sizing classifier separating, as overflow, fine material for treatment on slime tables; Callow screens preparing feed for sand tables. The undersize from finest screen is an enriched product suitable, after drying, to be directly smelted (pp. 196, 247).

CHAPTER VII

SIZING ; WATER-SIZING OR CLASSIFICATION

Behaviour of Particles in Water

The division of crushed material into classes according to size of particle, may be accomplished, if the particles are not too large, by submitting the material to water rising at a velocity sufficient to carry the finer particles upwards while permitting the coarser to fall. A consideration of the fall of particles in water is accordingly of prime importance.

Fall in Water.—A particle unsupported in water falls by reason of the unbalanced force represented by the excess of its weight over the weight of the equal volume of water displaced. Against this fall there is opposed the mechanical resistance of the water, this resistance being of two kinds. When the velocity of fall is relatively great, as it is with large particles, the action of falling must remove the water from the line of fall, impressing on this water velocity and movement which are finally absorbed in eddies ; such is "eddy resistance." When, on the other hand, the velocity of fall is small, as it is with very fine particles, eddy resistance is negligible, the controlling resistance being that due to the viscosity of the water, that is, to skin friction with the particle surface ; such is "viscous resistance."

In so far as the particles encountered in ore-dressing are concerned, it may be said that the point in the scale of size around which the respective provinces of eddy and viscous resistance overlap is about 0·15—0·25 mm., say 50—80 mesh. Departing from this critical size one or other of the two resistances becomes negligible, the other assuming control (Fig. 162).

Dealing first with the fall of larger particles and for convenience taking the sphere, the force producing acceleration from the position of rest, is the particle weight less the weight of an equal volume of water, that is $\frac{\pi D^3}{6}(\delta - \delta_0)\omega$, where D is the diameter of the particle, δ and δ_0 the densities of the particle and medium respectively, in respect to water as unity, and

ω the specific weight of water. Opposed to this force is the eddy resistance, which is given as $KA\frac{v^2}{2g}\delta_0\omega$, where K is a constant depending upon the shape of the falling particle, A the cross-sectional area of the particle normal to the direction of movement, v the velocity of that movement. In the case of a sphere, K from experiment may be taken to be 0·5 and the resistance becomes

$$0{\cdot}5\frac{\pi D^2}{4}\frac{v^2}{2g}\delta_0\omega.$$

Diameter of Particle.	Equivalent I.M.M. Screen.	Velocity determined by	
		Eddy Resistance $v=112\sqrt{d}$.	Viscous Resistance $v=700d^2$.
mm.		mm. per sec.	mm. per sec.
2·54	5 mesh	178	formula not applicable
1·574	8 ,,	140	
1·056	12 ,,	116	
0·635	20 ,,	89	
0·421	30 ,,	72	124
0·254	50 ,,	56	45
0·157	80 ,,	44 formula not applicable	17
0·106	120 ,,	36	8·0
0·063	200 ,,		2·7
0·042	300 ,, not made		1·2
0·025	500 ,,		0·44
0·016	800 ,,		0·17
0·006	1200 ,,		0·07

FIG. 162.

Terminal Velocities of Quartz Particles falling in Water.—The respective formulae contain constants determined from experiment, chiefly by Richards. It is probable, however, that the velocities obtainable from these formulae are a little too high (pp. 249, 276). Velocities and diameters are plotted in Figs. 164, 165.

The resistance, increasing with the square of the velocity, eventually becomes equal to the force producing acceleration, after which the velocity remains constant. This constant or "terminal velocity" is obtained by equating force and resistance, thus:

$$\frac{\pi D^3}{6}(\delta-\delta_0)\omega = 0{\cdot}5\frac{\pi D^2}{4}\frac{v^2}{2g}\delta_0\omega\ ;$$

whence, the diameter being in metres and the velocity in metres per second, $v^2 = 26{\cdot}16D\left(\frac{\delta-\delta_0}{\delta_0}\right)$ or $v = 5{\cdot}11\sqrt{D\left(\frac{\delta-\delta_0}{\delta_0}\right)}$;

or, using the more convenient units of millimetres and millimetres per second,

$$v=162\sqrt{\mathrm{D}\left(\frac{\delta-\delta_0}{\delta_0}\right)}.$$

More simply still when water is the medium :

$$v=162\sqrt{\mathrm{D}(\delta-1)}.$$

If instead of a sphere a cube be taken, the constant K is by determination 1·28, and the previous equation becomes

$$\mathrm{S}^3(\delta-\delta_0)\omega=1{\cdot}28\ \mathrm{S}^2\frac{v^2}{2g}\delta_0\omega\ ;$$

whence, the side S being in metres and the velocity in metres per second,

$$v^2=15{\cdot}33\sqrt{\mathrm{S}\frac{\delta-\delta_0}{\delta_0}}\ \text{and}\ v=3{\cdot}91\sqrt{\mathrm{S}\frac{\delta-\delta_0}{\delta_0}}\ ;$$

or, finally with millimetres in the place of metres,

$$v=124\sqrt{\mathrm{S}(\delta-1)}.$$

Actually, however, crushed particles are never so equidimensional as either spheres or cubes, but in falling present a greater cross-sectional area in relation to mass than either of those forms, with the result that they fall slower and at a rate which in respect to their average diameter is fairly well represented by the formula

$$v=\mathrm{C}\sqrt{\mathrm{D}(\delta-1)},$$

where $\mathrm{C}=50$—100, the lower figure for pronouncedly flat particles,
v = velocity in millimetres per second,
D = diameter in millimetres,
δ = density in reference to water as unity.

For this constant C, Richards[1] gives 87 for quartz and 100 for galena, these figures being the averages of many determinations. In the fact that the constant for galena is higher than that for quartz there is the suggestion that the constant increases somewhat with the density. Elaborately conducted experiments[2] upon perfect cubes and spheres have not, however, confirmed this suggestion. From the following considerations it will nevertheless appear that of two particles having the same ultimate terminal velocity, the denser accelerates quicker, reaching its terminal velocity earlier.

$$\begin{aligned}\text{Force}&=\text{Mass}\times\text{Acceleration},\\ \text{Acceleration}&=\text{Force}\div\text{Mass}\\ &=\pi\mathrm{D}^3(\delta-1)\omega\div\frac{\pi\mathrm{D}^3\delta\omega}{g}\\ &=g\frac{\delta-1}{\delta}.\end{aligned}$$

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[1] *Ore-Dressing*, p. 267.

[2] *Glückauf*, May 8, 1915.

This expression for acceleration is, of course, only true at the beginning of movement when resistance is practically non-existent, or has not fully arisen. Certain it is, however, that the denser particle gets ahead during the period of acceleration, cutting short that period (Fig. 163). Theoretically, the terminal velocity is only reached after infinite time; actually, with the size of particle ordinarily met in dressing, it is reached in a fraction of a second, this fraction increasing with the size of the particle. The relation of these periods of acceleration and full velocity are well shown in a graph with time as abscissae and space as ordinates (Fig. 163).

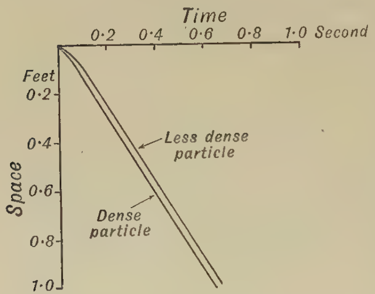

FIG. 163.

Particles falling in Water.—Diagrammatic plot of time against space. Of two equal-falling particles, the denser reaches the terminal velocity first, having a shorter period of acceleration. The parallelism of the graphs indicates that the two particles have the same falling velocity, that is, are equal-falling (pp. 252, 301).

The fall of particles so fine that viscosity of the medium was the controlling factor, was discussed when describing fractionation by elutriation. There, also, the general formula by Stokes, $v=\frac{2}{9}g\left(\frac{\delta-\delta_0}{n}\right)r^2$, was given, as well as its derivative when the medium was water, the viscosity was 0·01, and the velocity and diameter were in millimetres, $v=545(\delta-1)D^2$ (p. 206).

Such a value for the constant was confirmed by actual determinations by Richards,[1] these determinations indicating at the same time that the constant for quartz was again less than for galena, the respective figures being 424 and 631.

Reviewing these considerations upon the fall of particles in still water, and applying them to particles of the same mineral where $\delta-1$ would necessarily be constant, it is seen that the velocity of fall of very fine particles is proportional to D^2 and that of large particles to $\sqrt{D}$. In either case it increases with the size of the particle, and, accordingly, in the absence of difference in density a division of mixed material into sizes is possible.

With the coarser sizes of such material the velocity of fall is directly proportional to the square root of the diameter, and with the finer sizes directly to the square. Plotting velocity against diameter the resultant graphs would be two parabolas with axes at right angles (Fig. 164). Plotting the logarithm of velocity against that of diameter the resultant graphs would be two straight lines crossing at a point (Fig. 165). The previously-quoted determinations by Richards keep very well to these straight lines at the extremities, but towards the point of crossing they depart inwards to form a continuous curve. Along this curve neither viscous nor eddy resistance prevailed but each shared in determining the velocity of fall. For practical purposes the size corresponding to the centre of this curve may be regarded as the critical size. It is given by Richards as about 0·20 mm. for quartz and 0·13 mm. for galena.

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[1] *Ore-Dressing*, pp. 265-267.

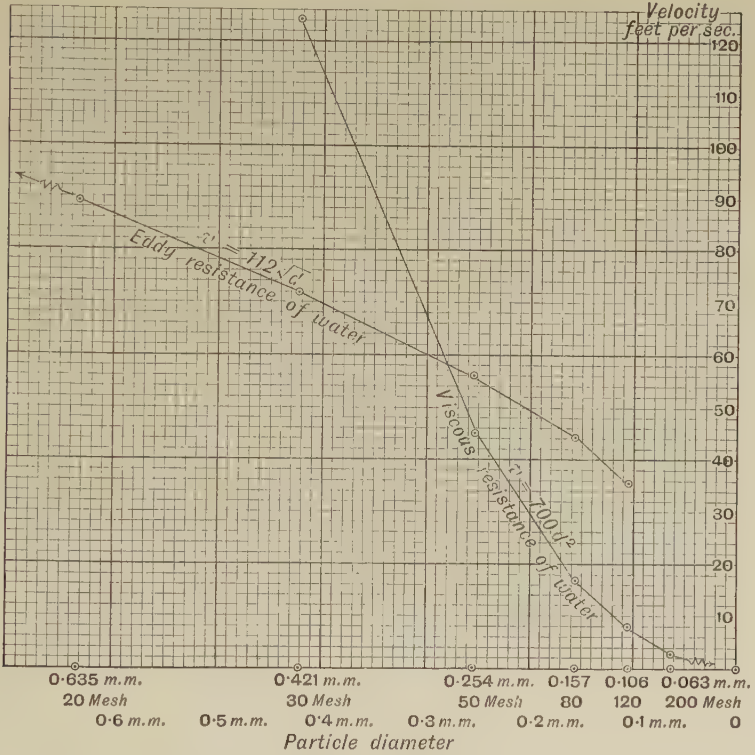

FIG. 164.

Quartz Particles falling in Water.—Simple plot of diameters against velocities, the actual figures being those given in Fig. 162. The curve represented by the formula $v=112\sqrt{d}$ (eddy resistance) is that of a parabola with a horizontal axis; that represented by the formula $v=700d^2$ (viscous resistance) is that of a parabola with a vertical axis (pp. 250-252).

Where difference of density exists, water-sizing, though still capable of separating particles which are coarser from those which on an average are finer, does not produce "equal-sized" classes, but classes made-up of particles having approximately the same velocity of fall in water, that is, it produces "equal-falling" classes.

In the knowledge of the conditions under which fall in a medium takes place, the discussion of the size relations of such equal-falling particles is possible. This discussion will be centred around particles falling under eddy- or turbulent resistance since the bulk of crushed material is of that class, and since also these relatively larger particles permit closer determinations.

It has been shown that the velocity at which such particles fall after a short period of acceleration is equal to $C\sqrt{D(\delta-1)}$. Equal-falling particles being those which have the same falling velocity it is evident that in the case of two such particles of different densities, δ and δ_1, and of different diameters, D and D_1 respectively, the relation $D(\delta-1)=D_1(\delta_1-1)$ or $\frac{D}{D_1}=\frac{\delta_1-1}{\delta\ -1}$ must obtain. Taking D_1 and δ_1 to pertain to galena of specific gravity 7·5, and D and δ to quartz of 2·6, the relation between the diameters would be $\frac{7\cdot5\ -1}{2\cdot65-1}$, or roughly 4, at which rate that

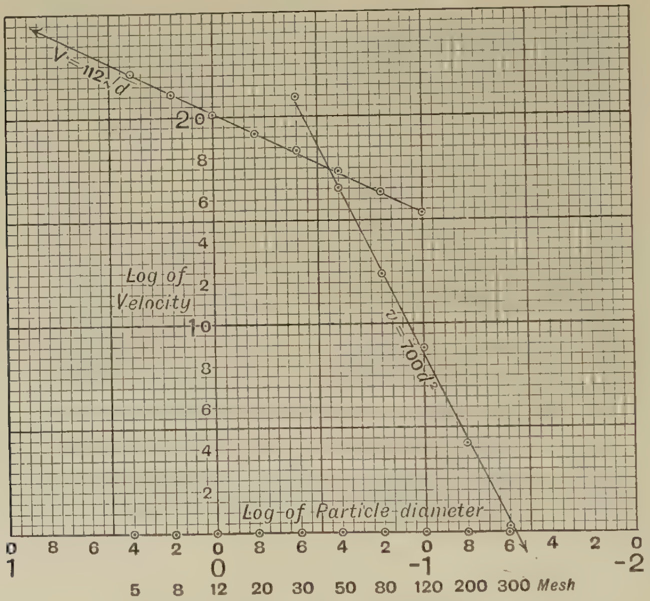

FIG. 165.

Quartz Particles falling in Water.—Logarithmic plot of diameters against velocities, the actual figures being those given in Fig. 162. In this plot the two curves $v=112\sqrt{d}$ and $v=700d^2$ become straight lines, intersecting at a size a little greater than 50 mesh. A curve joining these two lines instead of allowing them to intersect would be a more correct representation of the fall of particles approaching critical size (pp. 250-252).

between the volumes would be 64, and that between the weights 23. From this comparison it is seen that of two equal-falling particles not only has the denser particle a very much smaller volume but also a much smaller absolute weight.

In any other medium of density δ_0, the relation between equal-falling particles would be $\frac{D}{D_1}=\frac{\delta_1-\delta_0}{\delta\ -\delta_0}$. If, for instance, the density of the medium were 1·5, the "equal-falling ratio" would be $\frac{7\cdot5\ \ -1\cdot5}{2\cdot65-1\cdot5}=5\cdot2$, and the sizing effected would be still less perfect. Proceeding still further, if the density of the medium were equal to or greater than that of the lighter mineral the equal-falling ratio would be infinitely large; sizing would then be impossible, the only possible separation being one by density, the lighter mineral would float, the heavier would sink. On the other hand, if fall took place in air the equal-falling ratio would be diminished, becoming in the case of galena and quartz $\frac{7\cdot5}{2\cdot65}=2\cdot83$. Though this last ratio might indicate that, in air, sizing would be more perfect, actually, in the presence of so small a resistance the terminal falling-velocities would be so great that the chances of putting sizing into successful operation would be diminished; put another way, the difference in the falling velocities of large and small particle would be so much less relatively that sizing would be rendered more difficult.

Finally, if fall took place in a vacuum where $\delta_0=0$, the ratio would be

$$D\frac{\delta-\delta_0}{\delta_0}=D_1\frac{\delta_1-\delta_0}{\delta_0};$$

whence
$$\frac{D}{D_1}=\frac{\text{infinity}}{\text{infinity}}.$$

Here again, though the reduction of the size ratio might give the idea that, with density rendered of no effect, perfect sizing would be possible, in the absence of any resistance, actually all particles would fall at the same infinitely great terminal-velocity, and all chance of sizing would disappear.

The "free fall" which has been described demands that the falling particles should be so far apart as not to interfere with one another, and in such small number as not to occupy any considerable portion of the water area. Such conditions rarely occur in practice. Limitations of plant make it almost unavoidable that the particles fall close to one another, in which condition of proximity the descent of the larger particle is hindered more than that of the small particle, with the result that the size ratio of equal-falling particles is increased; the effect is indeed much the same as if the medium in which fall took place were no longer water, but a liquid having the greater density of the mixture of solid particles and water. This is the condition of "hindered fall" or hindered settlement; as will be seen later, this condition particularly obtains in certain classifiers known as hindered-settling classifiers, and on a jig bed.

Thus far the fall of particles in still water has been considered. In actual practice, still water is abandoned as impracticable, the material being submitted either to rising, falling, or horizontal currents.

The rising current is that most used. The pressure exerted by a particle upon the water when falling at a certain velocity, is equitably replaced by that exerted upon the particle by a stream of water rising at the same velocity. A particle will remain still and suspended in a rising current having a velocity equal to the terminal velocity of the particle, but would be lifted by a greater velocity, and fall through a lower velocity.

To effect sizing by a rising current, all that is necessary is that the material be brought to a plane across which the water is rising at the velocity capable of lifting the desired fine material to the overflow, while permitting the coarser material to fall. The classifier in which such a division is made needs, accordingly, no great vertical dimension, and there is the further advantage that the division would be made in a continuously-flowing pulp. Were fall in still water employed, great height of appliance would be necessary and it would be difficult to design an appliance to work other than intermittently.

It sometimes happens, however, as for instance in the descending side of a classifier and also in jigging, that particles fall in a descending current (p. 300). In such case the resistance to fall is weakened, and, since differentiation between particles is entirely dependent upon the discrimination of this resistance, the possibility of effective division into normal equal-falling products is diminished. Small size, low density, and large relative surface, the properties which cause a particle to settle slowly in still water, would cause a particle to move fast in a downward stream, whereas those other properties which promote rapid fall are precisely those which would diminish the effect of such a stream. It is conceivable, therefore, that a descending stream might be moving so rapidly downward that in their fall light particles would overtake heavier particles. Descending currents are accordingly detrimental to classification, and, under some circumstances, to jigging.

It frequently happens also that particles fall in a horizontal current, as for instance upon entry into some classifiers, settling boxes or pits, and in launders (Fig. 166). The vertical element of descent is then exactly the same as though fall took place in still water, but, in addition, there is a horizontal displacement in the direction of the stream. To this stream all the particles are subject during the time they take to fall; movement in response to the stream is governed by just the same factors as govern movement in response to gravity, with the result that the particles take a more-or-less diagonal line downwards, equal-falling particles keeping together. The heavier and more rapidly-falling particles being in the stream for a shorter time suffer less displacement, while the lighter and slower-settling particles, being carried forward along a flatter diagonal, are greatly displaced. It must be remarked, however, that should the bottom on which they fall be also in the stream, the friction of the water against that bottom introduces a new element of such great effect that the conditions sketched above are largely reversed, and the larger particles, which before the bottom was reached held an upstream position in relation to the smaller particles, now find themselves downstream. They find themselves in that position because, projecting into the swifter films of water, they are rolled forward, while the smaller particles lie still in the quiet film against the bottom. This action may be described as "film sizing"; it is essentially a stream effect (Fig. 166).

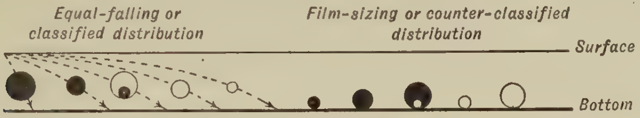
Equal-falling or classified distribution
Film-sizing or counter-classified distribution
Surface
Bottom

FIG. 166.

Stream Action.—Diagram. The illustration shows the equal-falling or classified distribution which takes place as mineral particles fall through a stream; and the redistribution which subsequently supervenes by movement along the bottom. The dark circles represent mineral and the lighter circles, gangue (pp. 257, 329).

Settlement in Water.—In ore-dressing, the maximum size of material submitted to water-sizing is limited, the large amount of water required to maintain the necessary velocity and the greater convenience of sizing coarse material by screens being the determining factors. On the fine side, however, there is no similar limitation; the material submitted includes suspensions and even colloidal solutions, the respective particles of which are in prolonged or permanent suspension. With such as these, to speak of falling velocity appears inapt and the expression 'rate of settlement' is more suitable. Moreover, in such settlement the aim is no longer a division of the material into different classes, but a separation of the solids from water.

With increasing fineness, particle surface, decreasing only as the square of the diameter, becomes relatively an increasingly greater factor than particle mass which decreases as the cube, till finally, surface energies, giving rise to electrostatic repulsion, and the inherent kinetic energy evidenced by the Brownian movement, delay or prevent settlement.

Such fine suspensions and colloids may, if desired, be precipitated by the addition of electrolytes known in this connection as "flocculators." "Flocculation" in slime settlement is the gathering of suspended or colloidal particles into aggregates, with the consequent separation of clear water. Such aggregates behave like particles of larger diameter; more particularly they fall and leave clear water above them. Hitherto, the "selective flocculation" of the mineral in the presence of dispersed gangue-particles has not been realized in ore-dressing, but the assembling of sulphide particles by a small amount of oil or infinitely-divided air in flotation concentration in effect must be regarded as such.

In relation to ordinary flocculation there are roughly three kinds of electrolytes; firstly flocculators, such as the inorganic acids and neutral salts, which, used in small amount, promote settlement; secondly, "deflocculators," such as the alkali salts, alkalis, and organic acids which used similarly bring about a dispersion of the separate particles; and finally, "protective colloids" which generally decrease flocculation, or at least prevent its development. This division is far from being invariably maintained, the effect of any particular electrolyte in any particular case being a matter for separate determination.

The flocculator most frequently employed in ore treatment is lime, this chemical being largely used to promote the settlement of slime for subsequent cyanide treatment. It is possible that, in effecting flocculation, lime is helped by incipient chemical precipitation, consequent upon reactions with soluble salts almost invariably present in waters available on mining properties. Other possible flocculators are sulphuric acid, alum, ferrous sulphate, while glue in very small amount has been suggested. Mine waters, other than those containing ammoniacal nitrogen, have usually a flocculating effect of themselves.

However desirable the assembling of disperse particles into the coagulated condition may be in preparation for a chemical treatment, it is not desirable when a mechanical separation follows. The floccules formed by the coagulation of disperse particles are gelatinous assemblies of heterogeneous individual particles with water, the mineral composition of the particles usually playing little part in the assembling; their gelatinous character is evidenced by the cohesion and shrinkage the settled mass exhibits upon drying. In a floccule consisting largely of gangue a mineral particle might find itself entangled, and thus withdrawn from any possibility of mechanical separation afterwards. The presence of a deflocculator would prevent such a happening. Deflocculators are, however, little used in ore-dressing except perhaps in connection with flotation concentration. Sodic carbonate is sometimes used, however, to keep kaolin in suspension while the accompanying fine particles of quartz and mica are given opportunity to drop out. Likewise, in making sizing analyses by elutriation it is necessary previously to deflocculate the suspension.

With deflocculated suspensions the particles fall individually to accumulate upon the bottom in a compact and dense layer, the suspension above continually growing more dilute but never completely clearing. With flocculated suspensions the whole assembly sinks together, there resulting not a gradually increasing layer upon the bottom but a mobile mass shrinking from the top, where clear water collects. Since the floccules contain water in their system, the shrunken mass is never compact as when the grains fall individually; its density depends upon the depth of water above.

Shrinkage settlement is simulated when the slime though not flocculated is thick, say containing more than 10 per cent of solid material. The presence of such an amount of solids increases the viscosity, preventing thereby the quick drop of individual particles, and to some extent the whole assembly moves down together, sweeping the supernatant water clear of suspended material.

In any case the rate of settlement is largely affected by the temperature. With increase of temperature there is decrease of viscosity and proportionately more rapid fall; as already stated, the viscosity of water varies about $2\frac{1}{2}$ per cent with each Centigrade degree of temperature within the normal atmospheric range (p. 206). Settlement is pronouncedly more rapid in summer than in winter.

CLASSIFIERS

Classifiers are water-sizing appliances. They are of two main types, namely, Surface Classifiers, wherein the sizing is effected at the water surface, by the water which brings the material to the classifier; and Hydraulic Classifiers, wherein it is accomplished in a restricted passage by fresh or added water introduced below. Surface classifiers are employed for finer material, the discriminating velocity being that which wells upward across the relatively extended surface at the level of overflow; hydraulic classifiers are used for coarse material, say above 80 mesh, the restricted passage permitting the requisite rising-velocity to be obtained with no great amount of added water.

Where it is no longer desired to divide fine material into sizes but to settle it, a third type of appliance known as a Settler is employed, which works on the same principle as the surface classifier, of which it may be considered to be a sub-type. Similarly, where it is desired to carry the hydraulic classification of coarse material well forward towards a density separation, a sub-type working under "hindered settling" conditions and known as the Hindered-settling Classifier has been developed. Under these four types classifiers may be described.

Surface Classifiers.—The ordinary surface-classifier is a pointed or pyramidal box placed in the stream with the point downwards; such a classifier is termed a Box Classifier or Spitzkasten (Fig. 167). Into it the pulp flows on one side and overflows on the other, inflow and overflow being generally on a level. With the overflow is carried such fine material as does not settle, while the coarser material passes out through a bottom or spigot discharge.

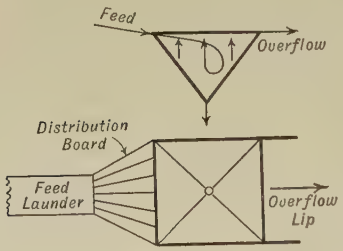

Fig. 167.

Box Classifier or Spitzkasten. —Diagram illustrating principle (p. 260).

The stream, entering on one side from a launder, usually has some horizontal velocity. This velocity, however, is largely absorbed in descending eddies, and the proportion of the inflow which passes directly to the overflow, even under conditions best suited to promote such a passage,[1] is small. It may accordingly be considered that the incoming stream all descends below the surface to rise again and overflow, the material overflowing with it being that possessing a falling velocity less than the velocity of ascent measured at the level of overflow (Fig. 162). In these classifiers, therefore, the overflow level is the sizing plane, or since the depth of water on the overflow lip is negligible, this lip being just awash, the sizing plane is practically coincident with the water surface. And the discriminating rising-velocity, v, is obtained by dividing the volume, Q, overflowing in unit time, by the area, a, of the classifier surface, or $v = Q/a$.

From that equation also, the capacity of a given classifier to effect a given separation may be determined; v and a then are known, whence Q is readily obtainable. But Q consists of the solid tonnage at a proper and known dilution with water, say a six-fold dilution; whence the solid tonnage is obtainable. The capacity of surface classifiers to effect a given separation is accordingly proportional to their area and, roughly, inversely proportional to the pulp dilution.

The material discharged at the spigot as underflow consists of particles which have rightly fallen, together with other lighter particles which have fallen owing to imperfections of the appliance. It is to effect convenient discharge of this fallen material that the pointed shape is given. The four sides slope inward at an angle which is not usually less than 60° from the horizontal. Where the classifier is square in plan, all four sides have the same slope; often, however, the dimension in the direction of flow is greater than across it and the slope of the two sides is steeper than that of the two ends.

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[1] Richards, *Ore-Dressing*, p. 238.

In consequence of their pointed shape these classifiers are deep and, except the bottom discharge be restricted, the pressure of water above will cause the discharge of too much water, the settled material will not pass out in properly 'thickened' consistency. By bending the discharge pipe upwards so as to deliver at a higher level, the pressure is reduced and the necessity for a restricted passage is avoided (Fig. 168). In this manner, while the chances of choking are eliminated, the consistency of the discharge can be readily maintained at the convenient relation of about 4—5 of water to 1 of ore. Such a goose-neck discharge is possible by reason of the fineness of the material, but were the material coarse or entirely sandy it would at once pack and choke.

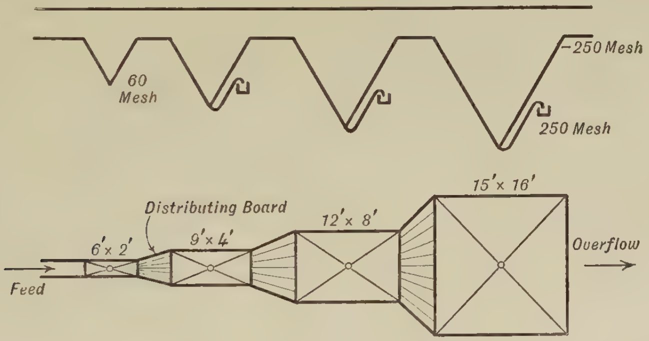


Fig. 168.

Spitzkasten Series.—Diagram of Rittinger series. Width of box increases in the geometric ratio of 2; length in the arithmetic ratio of 3. Goose-neck discharges at the finer spigots (p. 261).

To divide material into more than two classes a series of these boxes is required. Such a series was first elaborated by Rittinger, who, treating the matter empirically, took for his first box one having a length of 6 ft. and a width of 2 ft., a box which he considered to have a capacity of 1 cubic foot of pulp per minute per 0·1 ft. of width, or 20 cubic feet per minute altogether, this pulp containing 5—10 per cent of solids (Fig. 168). Following this box came three others expanding in width by the geometric ratio of 2 and in length by the arithmetic progression of 3 ft., the full series being:

	I.	II.	III.	IV.
Length	6 ft.	9 ft.	12 ft.	15 ft.
Width	2 ft.	4 ft.	8 ft.	16 ft.

At the inflow to the first and between each two of the remaining boxes, were distributing boards, these and the final overflow being practically on a level. Concerning the products, the conditions mentioned above would separate material of about 60 mesh in the first box and material finer than 200 mesh in the last box, the overflow containing but the very finest.

Such spitzkasten series are largely used in lead and zinc dressing on the Continent and have been used in lead and copper mills in America, the design and dimensions being varied to suit each particular case. Instead of successively doubling the width this dimension is, for instance, sometimes increased 50 per cent each time. Sometimes, also, instead of having completely separate boxes, a gradually widening and deepening trough, divided into boxes by transverse partitions, is employed (Fig. 169); or, again, one large box in which the intermediate partitions do not reach the level of inflow and final overflow. This last form is, however, perhaps better suited to settlement than to classification (Fig. 170).

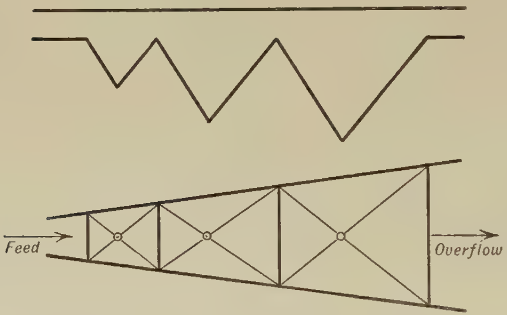


Fig. 169.

Spitzkasten Trough.—Completely-separate boxes along an expanding trough (p. 262).

Though these box classifiers may be considered to effect their separation in the welling ascent which follows the eddied descent, it is desirable to plan such an entry of the stream as shall carry the finer material well forward, clear of entanglement with the coarse. This is accomplished by inclining the inflow launder slightly, and keeping inflow and outflow on a level. The advantage thereby obtained, however, is not such as to warrant a similar inclination to any intermediate distributing-board; sometimes, indeed, as already pointed out, these boards do not exist, the stream passing gently over a partition into the next box. Baffles across the water surface, which are common with hydraulic classifiers, are out of place with these surface classifiers because of the obstruction they offer to the horizontal progress of the fine material. So also is the plunging entry which would take all the material, fine and coarse together, deep down into the dead zone where descent ends and ascent begins.

Under the best of conditions, however, the resulting classes are not sharply defined in respect to size, the settled coarse material always holding some fine material entangled, for which at the actual discharge there is nothing to bar the passage. Each succeeding class is, however, decidedly finer, while—when the material is friable, as chalcopyrite, or cleavable, as galena—the last product often contains more valuable mineral than the others, in which case not only has sizing been accomplished but some enrichment also. A further good point of these surface classifiers is that, while in every preceding operation—and be it remembered that the sizing of fine material is one of the end operations—water has probably been added, in these classifiers no such addition is necessary, but on the contrary the final overflow may sometimes be drawn off as clear water.

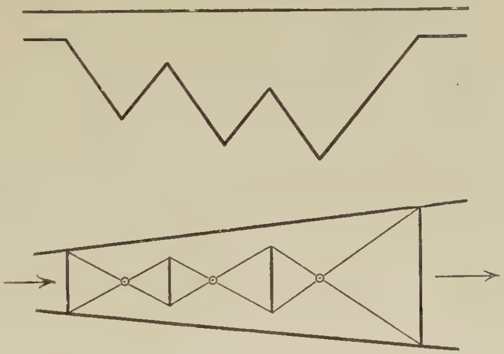


FIG. 170.

Spitzkasten Trough.—Incompletely-separate boxes along an expanding trough (p. 262).

The side feed and opposite-side overflow of box classifiers permit their ready assembly into series, while such classifiers are also convenient for the disposition and attachment of launders; in addition, the pyramidal shape is one easily constructed in wood. There is, however, this disadvantage, that the corners give opportunity for settled material to lodge, eventually to fall with disturbance to regularity in working.

The above disadvantage is avoided by adopting the conical shape. Cone classifiers do not, however, permit convenient assembly into series, nor are feed and discharge so simply disposed, the flow can hardly be maintained at a level throughout such a series. Nevertheless, where but one separation is required—as for instance that between sand and slime—the cone classifier may be preferred because of its smooth interior and its ready construction from iron plate. With such classifiers the feed is generally central, and down a short length of vertical pipe the lower end of which dips well below the water surface; such a manner of feed is termed 'sub-level delivery' (Figs. 171, 174). The overflow, on the other hand, generally takes place over the whole rim into a peripheral launder. Under such conditions of inflow and overflow, the descent upon entry and the subsequent ascent to overflow are easily pictured.

An illuminating variation of these conditions is embodied in the cone classifier designed by Mosher, in which the descent of the feed is peripheral and the ascent to overflow up through a central and vertically disposed enclosure, from the top of which a discharge lip delivers through the side of the cone (Fig. 172). Here it is clear that the classifying surface is that within the vertical enclosure to which the ascending movement is limited, entry taking place at the bottom. It is claimed that the particular advantage of this classifier is that it keeps the air-bubbles, which unavoidably enter with the stream, from rising in the ascending column and carrying with them particles which rightly should descend; and that in consequence a smaller classifier suffices.

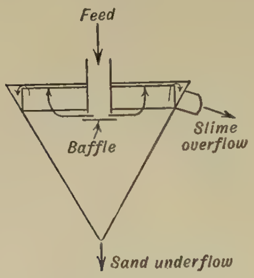


FIG. 171.

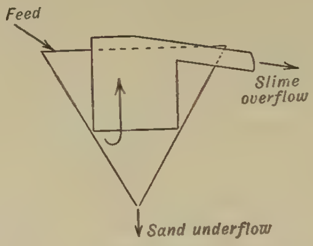


FIG. 172.

Cone Classifier (Fig. 171).—Central feed and peripheral overflow. Surface type (p. 263).

Cone Classifier (Fig. 172).—Mosher cone. Ex-central feed and central overflow (p. 264). A vertical plate down the centre of the rising column prevents vortex formation.

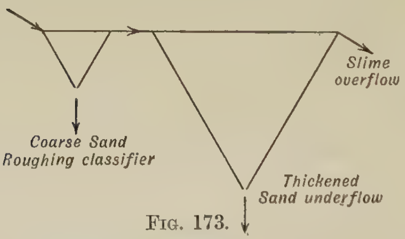


FIG. 173.

Roughing Classifier.—Diagram. A 'roughing classifier' takes out coarse sand preparatory to passing the material to a larger classifier for the separation of the slime. A 'sloughing classifier' removes by overflow most of the slime and the bulk of the water, leaving the underflow in a thickened condition (pp. 264, 265, Figs. 198, 205).

The simple cone works satisfactorily when the range of size is not great. It serves very well to 'slough off' slime from fine sand (Fig. 173). But when coarse sand is present, the great depth associated with the extent of surface necessary for the removal of slime would cause correspondingly great pressure at the discharge; using a simple cone, the choice would then lie between an adequate discharge-aperture passing too liquid an underflow, and a restricted discharge with the risk of a choke. This position would be relieved by taking out the coarsest material previously in a small shallow classifier, and for this purpose, a 'roughing' classifier is sometimes installed (Fig. 173). A more convenient means of securing a satisfactory discharge of relatively coarse sand while separating slime at the overflow, is to place a barrier or diaphragm just above the discharge to obstruct the straight drop through, a classifier so provided being known as a Diaphragm classifier. Finally, discharge may be accomplished by mechanically conveying or dragging the settled material upwards and out of the water along an inclined bottom, a classifier so operating being known as a Mechanical or Drag classifier.

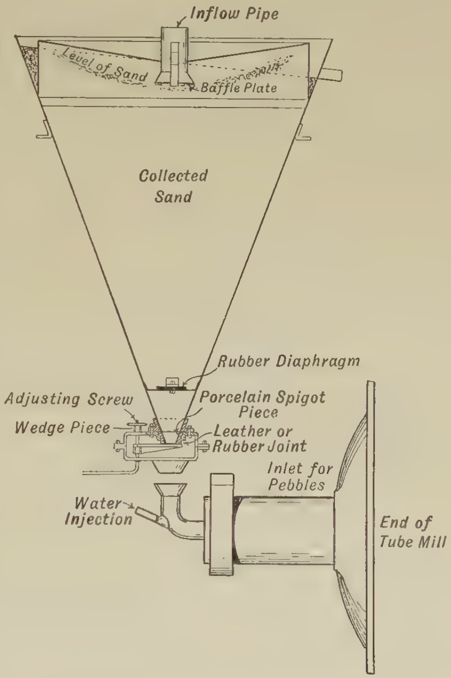


FIG. 174.

Diaphragm Classifier.—Caldecott cone. The illustration shows a diaphragm cone installed to separate sand for regrinding in a tube-mill. Delivery of feed is sub-level. The diaphragm is so effective in keeping back water from the underflow, that at the entrance to the tube-mill provision has to be made to add water to bring the feed to a proper dilution for the mill (pp. 263, 265).

The Diaphragm classifier appeared as the Caldecott cone, named after the inventor (Fig. 174). This cone is steep, the sides being at an angle of about 70° from the horizontal. Being circular there exists no particular side for inflow with natural outflow opposite, but the pulp is fed centrally down a cylindrical sheathing reaching below the surface, while overflow takes place all around into a peripheral launder. The coarse material sinks, but while the discharge is kept large enough to avoid choking, the clear way to it is obstructed by a diaphragm close above, this diaphragm being generally a circular plate fixed horizontally to leave between its circumference and the cone a narrow annular passage. In a cone 6 ft. in depth the actual spigot-discharge is about 1¼ in. in diameter, the diaphragm about 8 in. in diameter, and the annular space about ¾ in. across; with larger cones—some being 9 ft. in depth—the dimensions would be proportionally larger. This arrangement gives a very regular and thick discharge containing only about 30 per cent of water, instead of 60 per cent, the best possible with ordinary box-classifiers. The conditions which keep back water also keep back the fine material, and a cleaner underflow results. Discharge of the settled material is also so delayed that the classifier works almost filled with sand, only a basin-shaped depression remaining unfilled. Into this depression the feed falls centrally to sweep outward towards the rim, the coarse sand settling while the sufficiently fine material is carried to the overflow. In this outward movement, when with the classifier so full there is doubtless some streaming or launder action, the fine material finds protection between the large pieces.

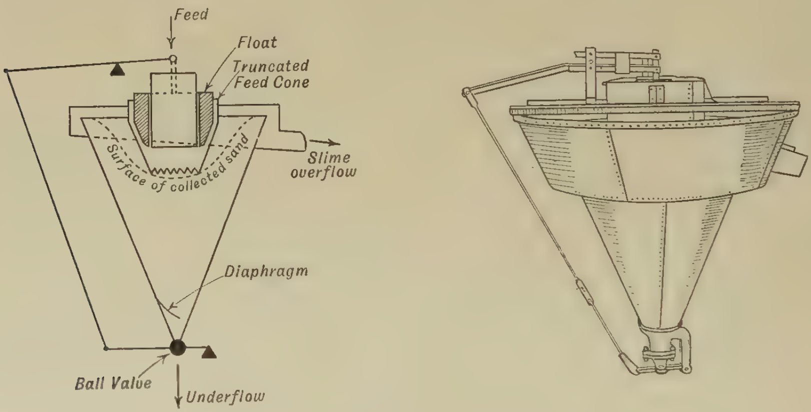


FIG. 175.

Allen Cone.—Diagrams illustrating principle and general appearance. This classifier adjusts itself automatically to varying conditions of feed. With greater rate of feed the float rises and by rods and levers the discharge is further opened (p. 267).

These diaphragm classifiers, while efficient and useful to separate sand for regrinding, do not achieve close sizing; of the slime overflow 10 per cent will usually be found coarser than normal, while the underflow will contain a similar percentage of fine material. Such a separation of sand would be satisfactory in preparation for regrinding. In any scheme of cyanide treatment, however, the final separation of slime from sand is better made in another classifier, probably one of the ordinary type, since any sand passing over with the slime will not receive an adequate treatment; similarly, in water concentration, the separation of the sand is best made by a hydraulic classifier, since any slime discharged with the sand will not yield to a treatment designed for the latter.

A similar cone, but one with the discharge controlled by a float within the feed sheathing, is the Allen cone (Fig. 175). With this cone, when, because of a coarser or bulkier feed, the collected sand rises so as to approach the bottom of the feed sheathing, the level of the water within that sheathing rises carrying the float with it, this float in turn through a lever and rods, further opening the discharge. This cone makes a good separation between granular material, however fine, and slimy material; and it is also used to remove the water from sand, an operation included under the term 'de-watering.'

The best known of the mechanical classifiers, the Dorr classifier, so named after the inventor, consists of a rectangular settling-box about 16 ft. long and 4—5 ft. wide, with a bottom so sloped that while the deep end is about 2 ft. 6 in. under water the other end is uncovered (Fig. 176). Into this box the pulp is fed from a short, perforated launder placed across the middle, the coarse material settling while the fine material overflows with the water at the deep end. As it settles upon the bottom the coarse material comes within the play of a series of rakes, attached about 5 in. apart to the under side of a framework which is so moved by a system of levers that the rakes keep close to the bottom as they move up the slope, and away from it as they move down. The length of this reciprocation is such that what one rake brings upward at one stroke is caught by the next rake at the succeeding stroke, the sand eventually being lifted over the uncovered end, far from the slime overflow. Before actual discharge and while lying uncovered, fresh water is applied to wash back any adhering slime. The number of strokes per minute depends partly upon the amount of sand but also upon the size and character of the material it is desired shall overflow. A common speed is ten strokes per minute, a faster stroke being given when it is desired to send larger and heavier material to the overflow, and *vice versâ*. The

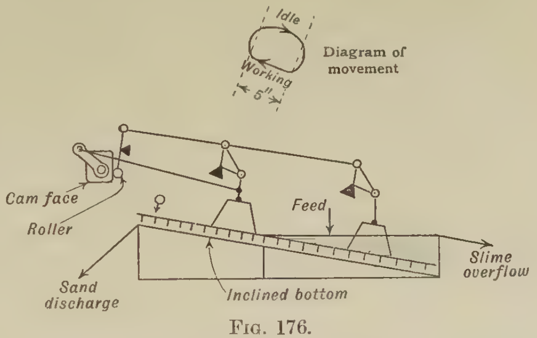


Fig. 176.

Dorr Classifier.—Diagram illustrating principle. Harmonic longitudinal motion given by crank and connecting rod; lifting and lowering motion operated through rods and levers from a cam on the same shaft as the crank. Stroke about 5 inches, or the same as the pitch of the rakes. The working stroke is the lower movement upwards; the idle stroke, the upper return movement (p. 267).

standard Dorr machine has two sets of rakes in the rectangular box, one set moving upward while the other moves downward (Fig. 177).

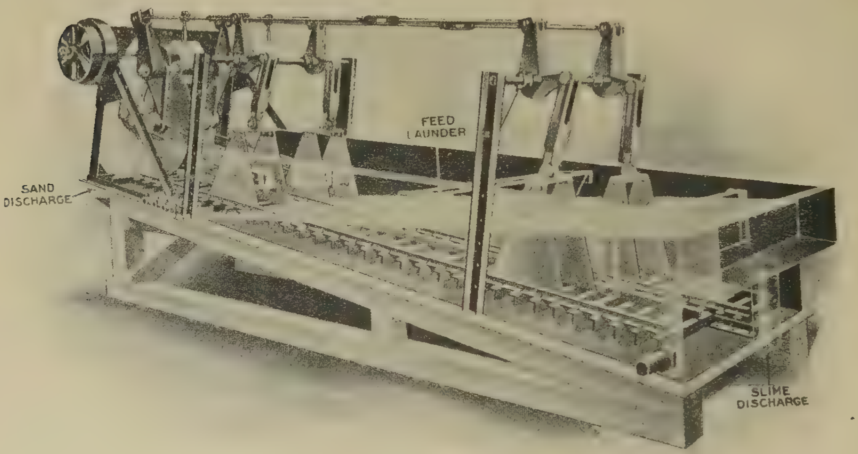


Fig. 177.

Dorr Duplex Classifier.—Phantom view. This illustration shows bell-levers, rods, connecting rod, cam and roller, crank, etc. The turnbuckles whereby the rods may be shortened or lengthened to adjust the stroke, are also shown (p. 268).

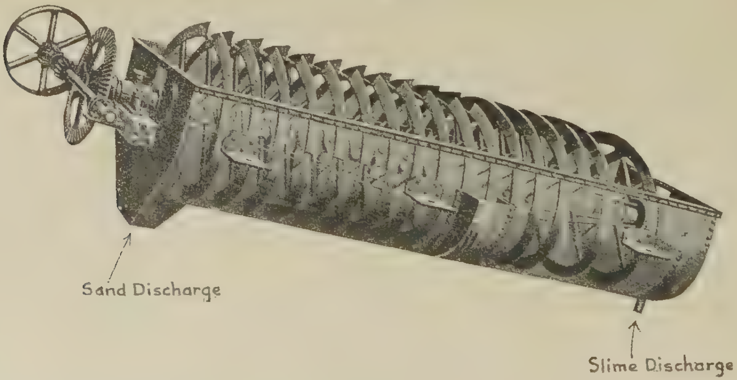


Fig. 178.

Akins Classifier.—Phantom view. The movement here is around the longitudinal axis, much like the log-washer. The flights are discontinuous in the upper half, but continuous and wider-spaced in the lower half. The trough or box of this classifier is semicircular (p. 269).

With these mechanical classifiers the products are better classified, the coarse product containing less than 10 per cent of fine material and the fine product less than 10 per cent of coarse ; moreover, the coarse product is accompanied by only about 25 per cent of water. In consequence, they are used not only in preparation for regrinding but sometimes as final arbiters between sand and slime. They have the further advantage that they require little height, discharge being practically on a level with the feed. The drag classifier also works comparatively undisturbed through irregularities of feed—the diaphragm cone, under excessive feed, fills higher and sends a coarser product to the overflow.

In the place of the reciprocating rakes of the Dorr classifier, blades attached to an endless belt to make an inclined scraper-conveyor are sometimes used ; or two parallel screw-conveyors, turning in opposite directions to squeeze the material between them as they force it upward ; or spiral flights attached to a framework revolving in a concentric trough (Figs. 178, 179). The Dorr, however, possesses the advantage that with it no parts moving on one another are submerged.

For the separation of very fine sand from slime an endless drag belt is sometimes used, the top portion of this belt forming the upward moving bottom of a very shallow box (Fig. 180). Upon such a belt and in shallow water, say, 2 in., the finest sand will quickly settle and be drawn up and out of the way, the slime overflowing. The sand so separated is relatively free from water and fine slime.

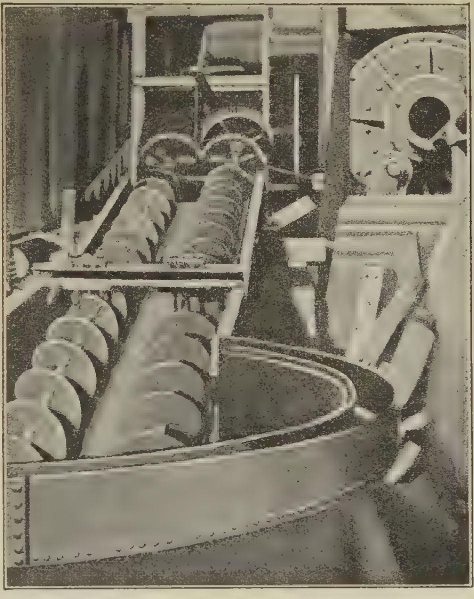


FIG. 179.

Ovoca Classifier.—General view. In this classifier the sand is pushed up the incline between two screw-conveyors, one right-handed and the other left-handed, which from the top revolve in towards one another. In this particular illustration the lower end of the trough is seen expanded into a bowl, giving the classifier settling, as well as classifying capabilities (p. 269).

Finally, there are several types of revolving scoop- or shovel-wheels, which, acting as de-waterers, lift the sand, giving opportunity for any entrained water to drain back, and opportunity for simultaneous washing with fresh water. In this connection it is worthy of remark that diminished water in the coarse product means freedom also from fine material.

The capacity of these mechanical classifiers, like that of all surface classifiers, depends upon the dilution of the pulp and the size of material it is desired to carry over; the greater the dilution and the finer the desired overflow, the less the capacity. In practice, when the separation is between fine sand and coarse sand, the capacity is about 30 tons per square foot of water surface per day, but when separating slime from sand it is only about 2 tons per square foot. This latter capacity is much less than that which would result from calculation, allowance having to be made for the fact that eddies and air-bubbles are liable to lift undue particles to the overflow.

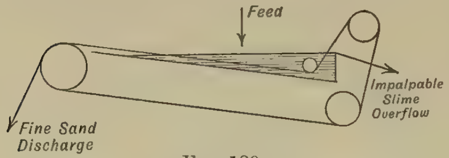


Fig. 180.

Drag-belt Classifier.—Diagram. The feed is carefully distributed; the average depth of water is about 2 inches (p. 269). (Cole, *Trans. A.I.M.E.* Vol. LV. p. 657.)

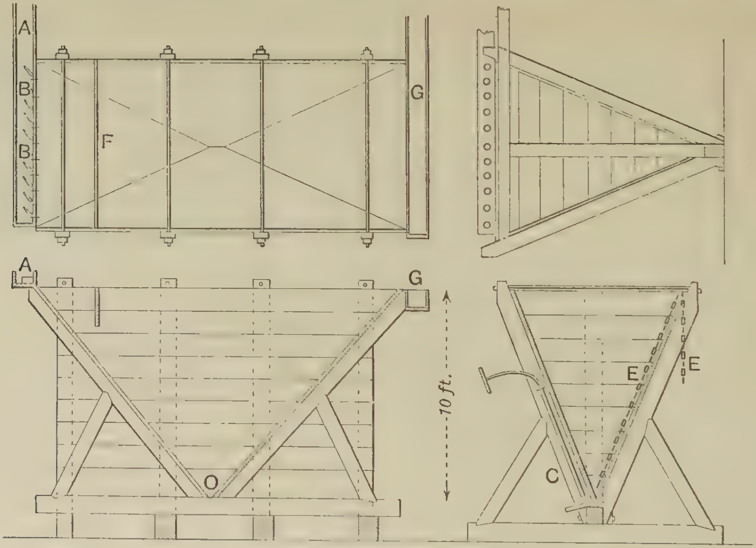
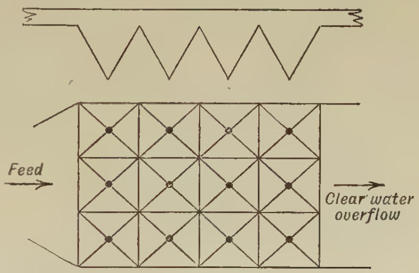


Fig. 181.

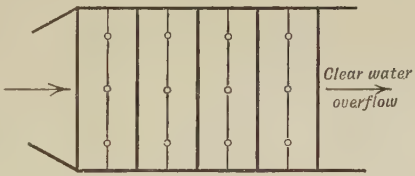
Box Settler.—A is the feed launder; B, the distributing buttons; F, a baffle; G, the clear-water overflow; E, a chain by which the discharge is regulated; and C, the discharge, which may either take place at the bottom, or by means of a gooseneck at some higher point (p. 271).

Settlers.—The appliances so named are those surface classifiers wherein the finest material settles from water, this latter overflowing. The material fed to them is that slime suspension which overflows the classifier proper. In the concentration of base-metal ores this suspension generally contains about 2—4 per cent of solids, such solids containing a higher proportion of alumina and a lower proportion of free silica than the original ore. In the preparation of precious-metal ores for cyanidation the classifier overflow will contain about 10 per cent of solids, the production of slime being greater. The settling of such material to the desired consistency is described as "thickening" and the appliance as a thickener.

The volume of inflow being great, a large surface is necessary in order that the movement upward to the overflow may not carry slime with it. Moreover, separating but one underflow product, there is no gradually expanding series towards the final overflow. At the entry, also, there may be a baffle to direct the whole stream downward, in that way favouring settlement. Otherwise, settlers are similar to surface classifiers; like them they may be divided into the Box, Cone, and Mechanical types.



Pyramidal Nest



"V" Nest

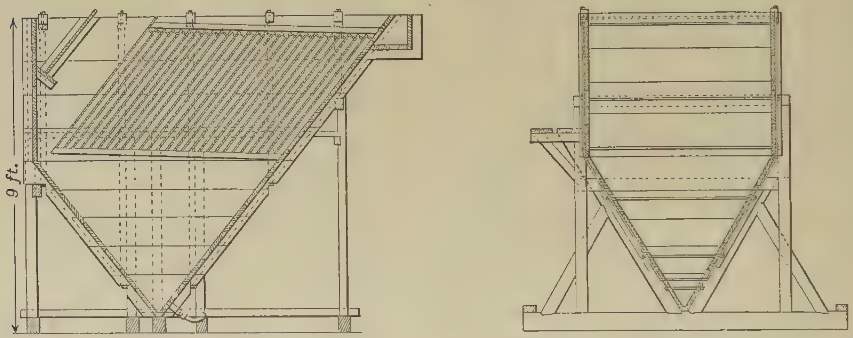
Fig. 182.

Settler Nests.—Diagram of Pyramidal and 'V' Nests. The spigots deliver to cross launders which in turn deliver to a side launder. The interior divisions do not quite reach to the surface of the water. The 'V' nest is more simple in construction (pp. 271, 272).

The Box Settler is generally a pyramidal box as before (Fig. 181). To facilitate downward progression of the material the slope of the sides must ordinarily not be less than 70°, and greater than 70° when a thick underflow is desired, because thick slime will accumulate and pack at such an angle. No great area can therefore be contained in a single pyramidal box without necessitating a height difficult to concede; accordingly, the required settling area has mostly to be obtained by a number or nest of such pointed boxes (Fig. 182). Where the collected material is subsequently all subjected to a common treatment—as in the cyanidation of slime—a nest of such boxes is convenient and readily assembled; but where this material has to be divided over many separate machines it is usual to have a number of smaller settlers, each installed in a position convenient to the machine it serves. Distribution of the dilute feed to separate settlers is more easily arranged than the subsequent distribution of the thickened underflow.

Instead of square pyramidal boxes, long rectangular V-shaped boxes placed across the direction of flow are sometimes used. Such a design avoids the making of many pyramids, discharge taking place through holes spaced at regular intervals. Several of such V-shaped boxes placed close together to take the pulp in succession constitute an effective settling nest (Fig. 182).

These pyramidal or 'V' settlers, in addition to the readiness with which they may be constructed of wood and assembled in nests, permit also the ready introduction of baffles as an aid to quick settlement, a subject discussed later (Fig. 183). The opportunity they provide for lodgment—and in this matter the V box is worse than the pyramidal—is, however, a point against them, and where separate settlers are permissible the cone settler is preferred. Cone settlers are similar to cone surface-classifiers except that they are larger (Fig. 184). They are fed centrally and below the level of the water surface, while the overflow takes place all round the rim into a peripheral launder. In the absence of corners there is not the same necessity for great steepness of side and an angle above 60° is not usually exceeded. This and the facility with which they are made in metal plate, are their advantages. Surface area being the factor determining the capacity of a settler, any advantage coming to cone settlers by reason of their relatively long length of overflow lip, even if worth mentioning, is small.



Section End View

Fig. 183.

Box Settler with Settling Baffles.—The solid particles in dropping away from the lower surface of each baffle leave clear water below that baffle. These particles fall on to the upper surface of the next baffle, down which they slide. Each baffle accordingly provides added surface from which settlement takes place (p. 272). (Ashcroft, *Trans. I.M.M.* Vol. XXII., 1913, p. 13.)

As with classifiers, the facilities for discharge largely determine the effectiveness of a settling appliance. So long as a thick underflow is not desired the progression of the settled slime to a position convenient for discharge is satisfactorily brought about by the slope given. The proportion of solids can be increased 2—3 times by a sloping bottom, that is to say, a pulp containing 5 per cent of solid will be thickened to 10—15 per cent in one operation, and, if undertaken, to 35 per cent by a second operation, this being about the limit possible in such settlers. Settled material of the greater thickness often desired for slime preparatory to cyanide treatment, or for flotation concentrate in preparation for shipment, or for residues preparatory to final discard, would build upon the slope and gradually fill the settling space. This is the position met by Mechanical Settlers.

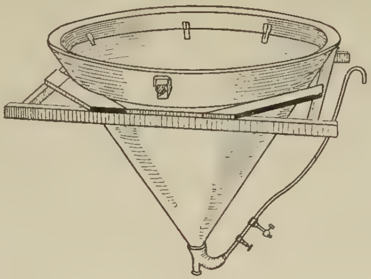
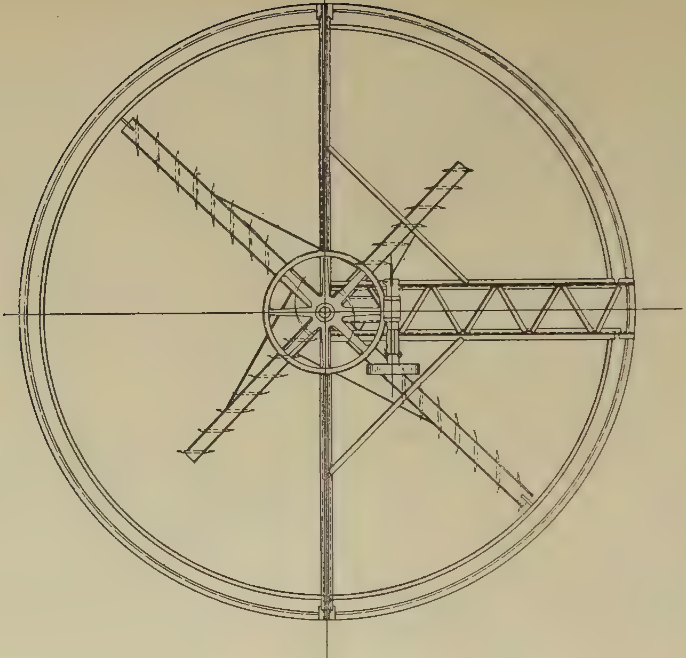


FIG. 184.

Cone Settler (Callow).—General View. The peripheral overflow, arranged inside the cone, leads to discharge at one point. The flexible goose-neck allows the bottom discharge to take place at a higher level. At the root of the neck compressed-air may be introduced to blow the settler clear when choked (p. 272).

Mechanical or Dorr settlers, being flat-bottomed, are provided with rakes which, while not moving at a rate to interfere with settlement, bring the material to the discharge before it has time to pack. These rakes are short ploughs attached to the underside of four radial arms, and so set, each as a short section of a spiral, that, upon revolution of the arms around the vertical spindle from which they are supported, the settled material is scraped from all over the bottom to the central discharge, this progression being assisted by sloping the arms about 10° downwards towards the centre (Fig. 185). Spiral ploughs, complete from periphery to centre, would effect the same conveyance but be more difficult to support.

The speed of revolution is naturally a slow one; with settlers of ordinary size, say 30 ft. in diameter, it is generally one-fifth of a revolution per minute for sandy material and one-eighth for slimy material; at such speeds the thickened discharge will contain 40 to 60 per cent of solids. Latterly, and in connection with large copper concentrators, much larger settlers have been employed, in which the rakes are no longer supported from a central spindle but from lattice girders bridging the diameter, and no longer operated from the centre but from a circular track around the periphery. With such as these, though the peripheral velocity may be more than maintained the speed of revolution is proportionally less.



Plan.

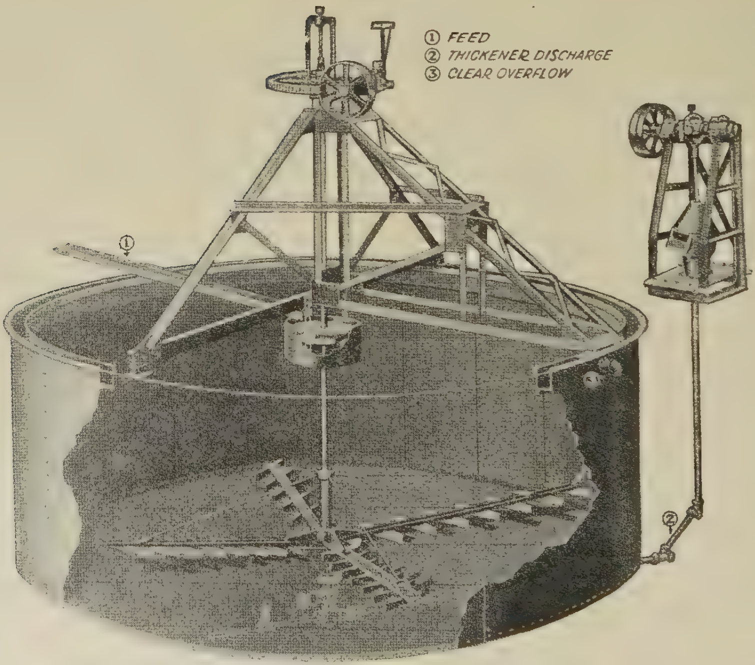


Fig. 185.

Dorr Thickener.—Plan and Phantom View. This appliance is used for thickening pulp preparatory to cyanide treatment or to slime concentration; to recover clear or nearly clear water from residues; or as a preliminary step in filtration when de-watering valuable material. A thickener of 30 ft. diameter requires less than one horse-power (pp. 273, 456).

Concerning settler capacity, since clear water can only form from the surface downwards, capacity depends upon the extent of surface; depth has no effect other than to increase the density or thickness of the discharge, 6 feet to 10 feet being usually found sufficient. Capacity will also depend upon the rate of settlement peculiar to the particular pulp, that is, upon the depth of clear water formed in unit time, a factor always capable of determination.

'Rate of settlement' depends upon many factors. It varies, firstly, with the physical and chemical characteristics of the solid in suspension. Speaking generally, aluminous material—probably because it is more friable but doubtless also because of some surface alteration—takes longer to settle than siliceous material. Of particular ore-minerals, Shellshear[1] found that cassiterite and wolframite were slow-settling; chalcopyrite and bornite in an intermediate class; and blende, galena, pyrite, fast-settling. And among gangue-minerals Shellshear put felspar in the slow-settling, quartz in the intermediate, and calcite, garnet, and fluorite, in the fast-settling classes, respectively.

Secondly, the rate of settlement depends upon the flocculating effects either of impurities in the water or of agents added. Inorganic acids, and particularly sulphuric acid; lime; neutral salts, and particularly calcium chloride, magnesium sulphate, alum, and iron salts, all promote flocculation and settlement. The alkalis, on the other hand, are ordinarily deflocculators, though when present in small quantity they have a flocculating effect. Finally, many organic acids, and particularly tannic acid, are deflocculators in all conditions of concentration, promoting the formation of permanent suspensions. Mine waters, as well as the surface waters available for dressing purposes, usually have a flocculating effect by reason of neutral salts in solution, though occasionally when ammoniacal salts are contained they may have an opposite effect.

Though flocculated material settles readily and leaves clear water above, it always remains a mobile and voluminous sediment, necessitating care that it be not caught and drawn away to the overflow. Pronounced floccules such as those resulting from the addition of a flocculator, are only of advantage when the collected material is to be treated chemically, as in cyanidation; in ore-dressing the assembly of different materials into floccules would render mineral separation more difficult, and only selective flocculation, were it possible, would be useful.

Settlement depends also upon the temperature; rise of temperature decreases the viscosity, thereby increasing the rate of fall. Between winter and summer in many places the capacity of a plant for effective settlement may vary as much as 20 per cent. Rate of settlement also varies with

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[1] *Aus. I.M.E.*, Dec. 31, 1912

24 hours

the density of the pulp. A dilute pulp is generally considered to settle quicker than a thick pulp, the particles of the former settling freely and individually, the latter pulp settling by shrinkage *en masse.* That is not, however, to say that a given amount of solids would be quickest settled by diluting the pulp, but only, that of two pulps one thick and the other thin, clear water could be drawn at a quicker rate from the latter than from the former.

With the rate of settlement ascertained by test, the capacity of any settler, or, alternatively, the size of the settler necessary for a given volume of pulp, can readily be determined. Rate of settlement takes the place of the falling velocity in the equation $v = Q/a$, where Q, though more precisely the volume overflowing, may be taken to be the volume entering.

In practice it is found that for thickening flocculated slime preparatory to cyanidation an area of 5 sq. ft. of settling surface is required per ton settled per 24 hours if the slime be siliceous, and 7—10 sq. ft. if it be aluminous. The incoming slime in such work has usually a consistency of about 10 : 1, and the underflow an average of about 2 : 1, denser for the siliceous slime and less dense for the aluminous. For every ton of solid treated per day, about 8 tons or 256 cubic ft. of clear water would pass to the overflow across an area of, say, 7·5 sq. ft., that is, 0·003 cub. ft. per second, equivalent to a rising velocity of 0·12 mm. per second, which, it will be observed, is about capable of lifting a quartz particle of 800 mesh (Figs. 124, 162). In ordinary water-concentration, though the consistency of the inflow is decidedly less, and rather of the order of 20—40 to one, it is nevertheless usual to allow much the same settling area per ton settled per day as in cyanidation work, and the rising velocity is correspondingly higher; it is, in fact, commonly of the order of 0·4 mm. per second, equal to lifting particles of 500 mesh. For the settlement of fine flotation-concentrate, seeing that any solid escaping would mean the loss of valuable material, the allowance of settling area is more liberal, being 10—40 sq. ft. per ton of concentrate settled per 24 hours.

An interesting method of increasing the effective surface of a classifier is to place just below the surface a nest of inclined baffles.[1] From the underside of each of such baffles, the solids fall and clear water separates. The theoretical increase of surface is consequently the product of the total area of the baffles and the cosine of the angle of their inclination, and the separation of reasonably clear water will be increased practically to that extent; a simple laboratory-test in an inclined glass cylinder gives a very good idea of this possible increase (Fig. 187). These baffles are generally placed across the stream and dipping downwards in the direction of the flow. In time they get fouled and clogged so that they are not often included in a permanent installation. The Dorr tray-thickener takes advantage of the same principle, its inclined baffles or trays approaching the horizontal (Fig. 186).

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[1] Stevens and Degenhart, *Monthly Journal Chamber of Mines, Western Australia*, March 1911; Ralph Hayden, *Trans. A.I.M.E.*, 1914, Vol. XLVI. p. 239; Watt, *Trans. A.I.M.E.* Vol. LVII., 1918, p. 441.

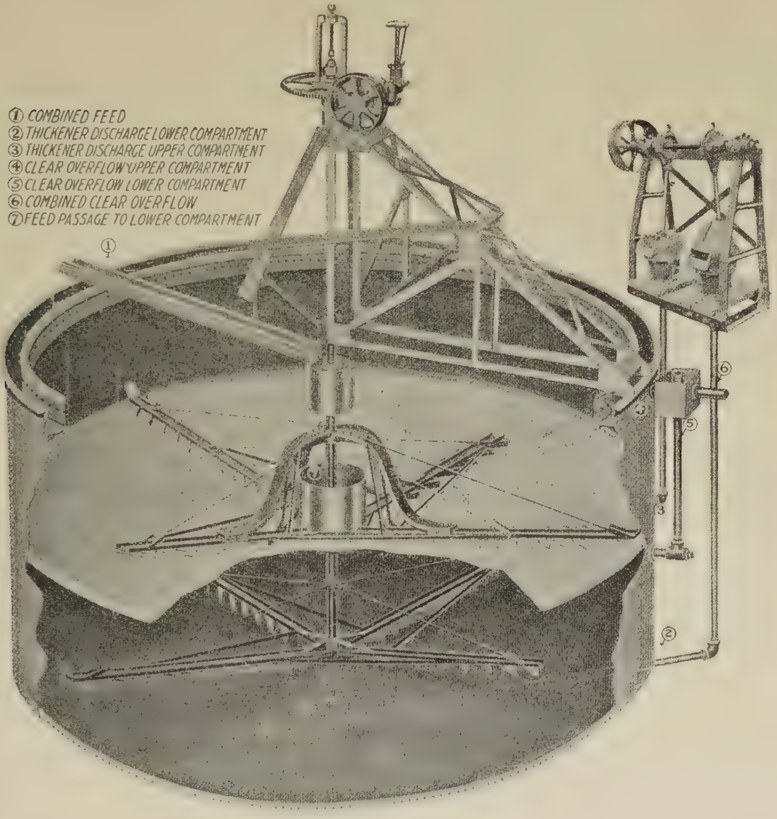
① COMBINED FEED
② THICKENER DISCHARGE LOWER COMPARTMENT
③ THICKENER DISCHARGE UPPER COMPARTMENT
④ CLEAR OVERFLOW UPPER COMPARTMENT
⑤ CLEAR OVERFLOW LOWER COMPARTMENT
⑥ COMBINED CLEAR OVERFLOW
⑦ FEED PASSAGE TO LOWER COMPARTMENT
①
⑥
⑤
3
②

Fig. 186.

Dorr Tray-thickener.—Phantom View. This thickener gives a large settling area in a comparatively small space. Clear solution or water is taken off from the upper layers of each compartment. The material settled on each tray is separately discharged, being generally drawn off by the suction of a pump or air-lift. In the centre of the upper trays is a feed passage to the lower compartments. As many as five trays have been installed in one deep tank (pp. 277, 456).

Where flocculators are used or the pulp flocculates of itself, the supernatant water is generally perfectly clear, the settled pulp appearing like a denser and immiscible liquid beneath. When, on the other hand, more or less permanent suspensions are formed and flocculators are not used, the overflow even after prolonged settlement is somewhat turbid. The amount of solid it contains is, however, so small as not to warrant the expense of further settlers of the types described, but if desired it can be settled in pits.

Settling pits as applied to ore-dressing are generally rectangular in plan and 6—15 ft. in depth, with inflow at one end and outflow over a gate at the other. While settlement is proceeding, this gate is built-up with slats to keep the pit full, the slats being removed subsequently one by one, when it is desired to draw off the supernatant water, and, in turn, the settled slime. Discharge of the latter is accomplished by water purposely entered at the feed end in amount suitable to give the resultant pulp the consistency desired. During the time one pit is filling, another is being discharged, and a third is usually standing quiet while the water is clearing and the settled slime consolidating. Such pits are sometimes used to thicken ordinary slime. For such work they are crude, not only because they are intermittent in action, but also because from end to end a classification develops, the coarser material settling at the feed end and the finer at the discharge, with the result that the product they deliver is irregular, slimy at one time, sandy at another. Settling pits or tanks worked intermittently possess advantages when a sediment of very thick consistency is desired.

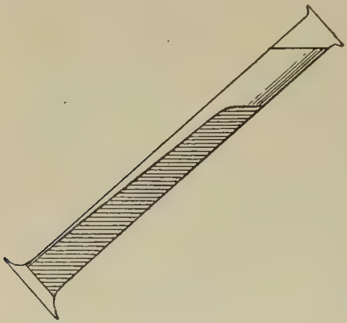

Fig. 187

Inclined Baffles as an Aid to Settlement.—Settlement in an Inclined Jar. If a suspension of slime be settled in a glass jar laid in an inclined position, clear water will form all along the upper side of the jar, and then move upwards in a stream to join the clear water forming directly under the free surface of the water. Rate of settlement being determined by the depth of clear water collected above the settling suspension, it is obvious that the influx of this rising stream will increase that depth beyond what it would have been had clear water only formed under the free surface. The clear water rises because it is lighter (p. 276).

Hydraulic Classifiers.—When describing the working of surface classifiers it was stated that some fine material, which rightly should overflow, finds its way to the bottom, where, with nothing to bar the passage, it is discharged with the underflow. So long as no great range of size exists in the material being classified, no great harm results. When, however, the range is such that this fine material would escape the effort of the separating machine set to treat the underflow, surface classifiers become out of place. This is the position met by hydraulic classifiers, the feature of which is the introduction of fresh water below, to make the underflow of unduly fine material impossible. In addition, therefore, to the pointed box or cone—such being then known as the ‘pocket’—hydraulic classifiers have at the bottom a ‘sorting tube or orifice’ through which fresh water passes in quantity sufficient to create the rising velocity necessary to turn back the fine material, a velocity determined by the terminal velocity of the particle which it is desired shall not pass, being somewhat greater than that velocity (Fig. 188).

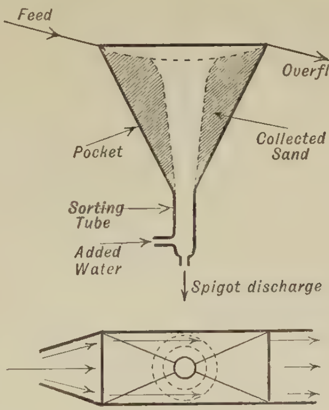

FIG. 188.

Hydraulic Classifier, Box Type.—Diagram illustrating principle of action. The sand collected in the pocket is supported there by the force of the rising stream (p. 279).

With its passage to the underflow thus barred, the fine material must be lifted to the overflow. The water available for that purpose is now the water inflowing with the feed increased by that rising through the sorting tube, and the surface of the classifier should be so dimensioned that the upward velocity of this increased amount of water across the level of the overflow should be the same as that in the sorting tube. But if complete provision is to be made that the material refused descent through the sorting tube should be lifted to the overflow, the same conditions in respect to uprising velocity must also obtain throughout the pocket. This final necessity is met automatically by the classifier itself, which permits sand of about the limiting size to collect in those parts which are out of the stream, till in the remaining passage the requisite velocity obtains. Roughly then, what happens, is that in effect the sorting tube is prolonged upwards by the collecting sand till it becomes merged in the surface of the classifier; in other words, the classifier fills till only a basin-shaped space remains, into the bottom of which the sorting column enters.

In the shallow water of this basin there is more of horizontal than of vertical movement, and more of streaming action than of direct lift. Whatever the shape of the actual sorting tube, the portion of the stream bed which it commands is circular. Particles taking the line across the centre of that circle will have every possibility of being properly sized, but particles passing to the sides, particularly if the classifier be wide, will have some chance of passing unchallenged (Fig. 188). Accordingly, hydraulic classifiers, with the surface unbaffled, give an overflow which contains some coarse material which rightly should be in the underflow.

To prevent coarse material so passing to the overflow, hydraulic classifiers are generally provided with a baffle-board placed across the stream to direct all the material downward, forcing it thereby to enter the sorting column on the other side of the baffle (Fig. 189). In classifiers so provided the sand collects as before, and the free way around the lower edge of the baffle becomes constricted, requiring then a certain pressure of the pulp to keep the way open. This pressure is provided by a difference in level between inflow and overflow, a difference sufficient to burst any choke which might temporarily form and therefore known as the 'bursting head.' On the rising side of such classifiers, and presuming the classifier is of box or conical shape, sand collects to form an upward passage of roughly uniform section, and, unless the baffle be deep enough, there still remains some chance of coarse material being streamed to the overflow. Accordingly, a deep-pocket classifier permits a better separation than one with a shallow pocket (Figs. 189, 195).

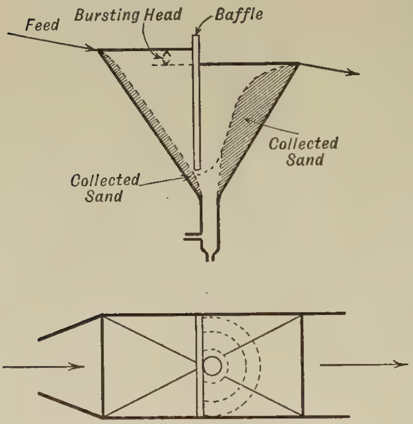

Fig. 189.

Hydraulic Classifier, with Baffle.—Diagram illustrating constant conditions of working. Deep pocket (p. 280).

Seeing that where the classifier expands in section upward, sand collects to keep the sorting passage uniform and the velocity constant, the natural design for a classifier would be one with tubes for the descending and rising columns, tubes of such a size that they would be swept clear of collecting material. A pointed tube is indeed the ideal classifier (Figs. 190, 191). Rittinger carried out this idea of uniform section in his Spitzlutte, by having as baffle a short triangular prism set with one edge downwards in a similarly shaped box (Fig. 192). By raising or lowering this triangular baffle the velocity of the current could be adjusted to suit varying circumstances, but with the baffle in any one position the velocity roughly constant from top to bottom.

A similar equality of section would be obtained by a vertical baffle if the sides of the classifier, back and front, were vertical; or by a flatter cone inside a steeper cone if the uprising stream were made to pass between the two cones (Fig. 193).

The foregoing considerations have been centred around conditions in the pocket. Proper conditions in the sorting tube and at the spigot discharge are equally important.

Water rising in the sorting tube or across the sorting orifice must properly command the passage. A short length of circular pipe is a common design, the added water entering this pipe either from the side or from a pressure box below. When the length of this pipe is too short some fine material will be taken down by eddies; on the other hand, when it is too long the vertical dimensions of the classifier and the pressure upon the discharge are unduly increased. This latter consideration has led to the adoption of a simple orifice in a plate inserted between the pocket above and the pressure box below. Of the two designs, the sorting tube is best suited to a pointed pocket, and the plate orifice to a pocket with width constant from top to bottom, the orifice being then conveniently rectangular (Fig. 194).

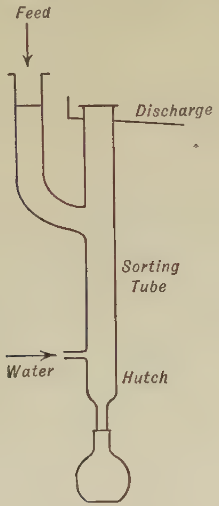

Fig. 190.

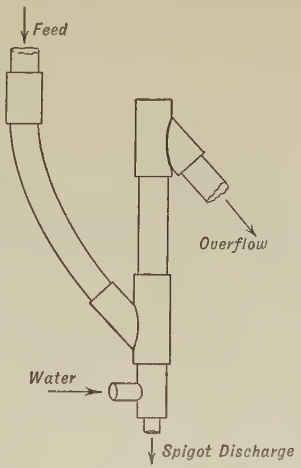

Fig. 191.

Laboratory Tube-classifier (Fig. 190).—Diagram. In this classifier since all the water is introduced below, the sorting column has the same section throughout pocket and sorting tube, and nothing collects in it (p. 280).

Pipe Classifier (Fig. 191).—Diagram. This classifier can be made of ordinary wrought-iron tubes and fittings. The feed must arrive under a pressure great enough to lift to the overflow. Such a classifier is only suitable for limited quantities, but the separation it makes is good (p. 280).

The area of the passage will depend upon the amount of material expected to pass, a diameter of 3 inches being common; in turn the area of the passage will determine the capacity of the classifier upon a given material, and the amount of water uprising. Not more than one-third of the sectional area of that passage should be considered as available for the stream of descending particles, the remaining two-thirds being for the ascending water. The velocity at which those particles descend may be taken to be the falling velocity of the average-sized particle diminished by the rising velocity of the water.

Below the sorting tube or orifice comes the pressure box from which, and generally from the bottom, the underflow of fallen material takes place under pressure of the column of pulp above (Fig. 193). In this underflow the water which enters the classifier with the feed does not take part, the underflow water being fresh or added water. Accordingly, the added water necessary to work hydraulic classifiers must both maintain an upward rising current in which sorting or discrimination is accomplished, and discharge the underflow. The amount of this water uprising will depend

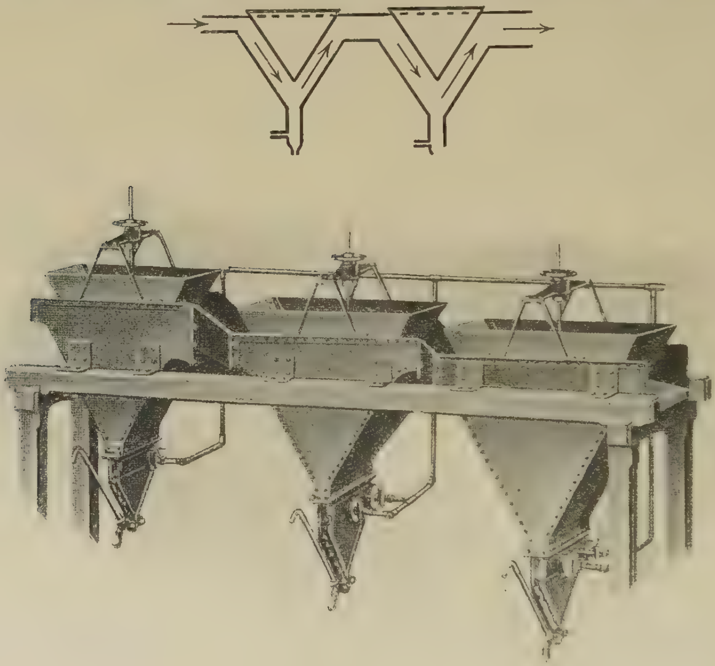

Fig. 192.

Rittinger Spitzlutte or Pointed Tube.—Diagram and General View. It is noticed in the illustration that the two last Spitzlutte of the series, being larger than the first, have each two entries for the water, whereby the long slot which constitutes the sorting tube is more regularly served. Below this water entry this slot contracts, till at the bottom it is brought to the point where a goose-neck discharge is attached. The adjustment of the height of the internal prism, and consequently of the width of the classifier passages, is made by means of a hand-wheel on a central rod (p. 280).

upon the area of the sorting passage, which therefore must not be too large; the amount discharging will depend upon the area of the discharge, and this, to prevent choking by particle interlock, should be about one-third that of the sorting passage. Roughly, it may be considered that the added water is about equally divided between that passing upward eventually to overflow, and that passing out with the discharge.

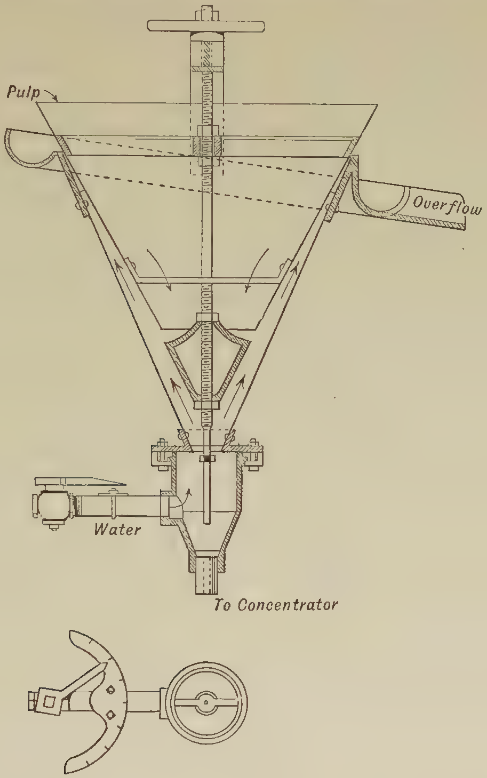

Fig. 193.

Baffled Cone-classifier.—Section. In this illustration the fresh water enters a pressure box out from which it passes upwards around a small internal cone into the classifier pocket. The vertical position of this internal cone is adjustable by means of a hand-wheel on a rod to which the cone is attached. Accordingly, the sorting tube—this being the passage between the main and internal cones—is capable of being adjusted to vary the products. Adjustment is also possible by means of an index cock upon the fresh-water service. The greater portion of the pocket is occupied by a feed cone into which the pulp enters and by which it is directed down to the sorting passage. The annular space around the feed cone is the continuation of the sorting column; the feed cone being flatter than the main cone, increasing diameter is compensated and the cross-sectional area of the sorting column maintained fairly constant (pp. 280, 282, 284).

Regulation of the amount of fresh water is made by a cock or valve on the supply (Fig. 193). Sometimes a cock is also used to regulate the discharge, but the narrow aperture which a cock gives when approaching the closed position, brings liability to choke. It is better to regulate the discharge by changing the actual aperture for another of different diameter. If the discharge be altered, the amount of water passing up the sorting tube also suffers alteration unless the supply be suitably adjusted.

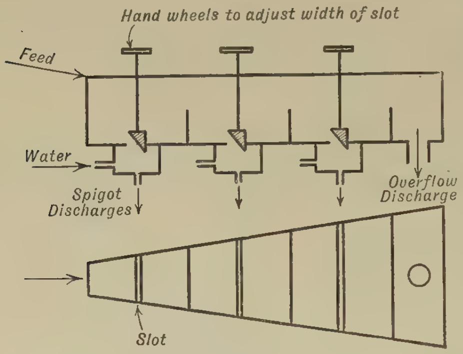

FIG. 194.

Slot Classifier.—Diagram of Yeatman Classifier. Slots in the classifier bottom serve as sorting tubes; the width of the slot is adjusted by raising or lowering a wedge-shaped block (p. 281).

Where the two products of underflow and overflow are not a sufficient division, two or more hydraulic classifiers may be placed in series, each spigot delivering material suitable both in size and amount for the machine or machines it serves. In such a series the succeeding classifiers are successively larger, partly to give the smaller velocities necessary for fine material, partly to accommodate the fresh water rising in each sorting tube. As each classifier demands a certain difference of level between inflow and outflow, the total height required becomes a consideration. The ordinary single classifier is generally a deep-pocket classifier, depth ensuring closer work. It is not often, however, that such deep-pocket classifiers are assembled in series. Rather is depth sacrificed to general convenience by arranging shallow pockets as depressions at regular intervals in a trough or launder (Figs. 195, 196). Or, the pockets

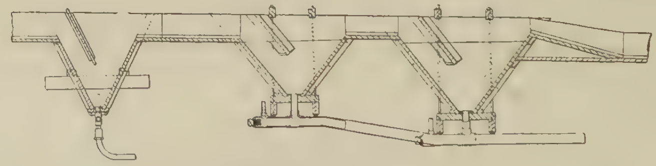

FIG. 195.

Shallow-pocket Classifiers in Series.—Longitudinal Section. In the two last pockets the baffle is shown dipping but little below the level of discharge. Undoubtedly, therefore, some coarse particles would be streamed to the overflow. On the other hand, a shallow pocket requires less height and less water issues with the discharge, water being thereby saved (pp. 280, 284).

may be entirely abandoned, the sorting tube or orifice being attached directly to the launder to form a launder classifier (Fig. 197). Launder classifiers are, however, not entirely satisfactory, since the stream carries much of the coarser material over and beyond its proper sorting column; but requiring so little height they are sometimes adopted.

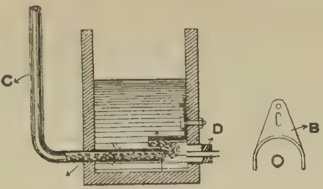

FIG. 196.

Shallow - pocket Classifier (Calumet Classifier).—Section across Pocket. In the illustration the fresh water is delivered by the pipe C into a semi-circular shield B, out from which it escapes, and escaping, gives rise to the discriminating current. In the side of the box and opposite to the clear-water entry is the spigot discharge out through which pass those particles heavy enough to have made their way through the current (p. 284). (Commans, *Proc. Inst. C.E.* Vol. CXVI.).

Hydraulic classifiers are sometimes worked in conjunction with surface classifiers, a classifier of the latter type removing the bulk of the slime before the hydraulic classifier is entered (Fig. 198); or, the two types of classifier may be combined in one continuous series, the hydraulic pockets preceding

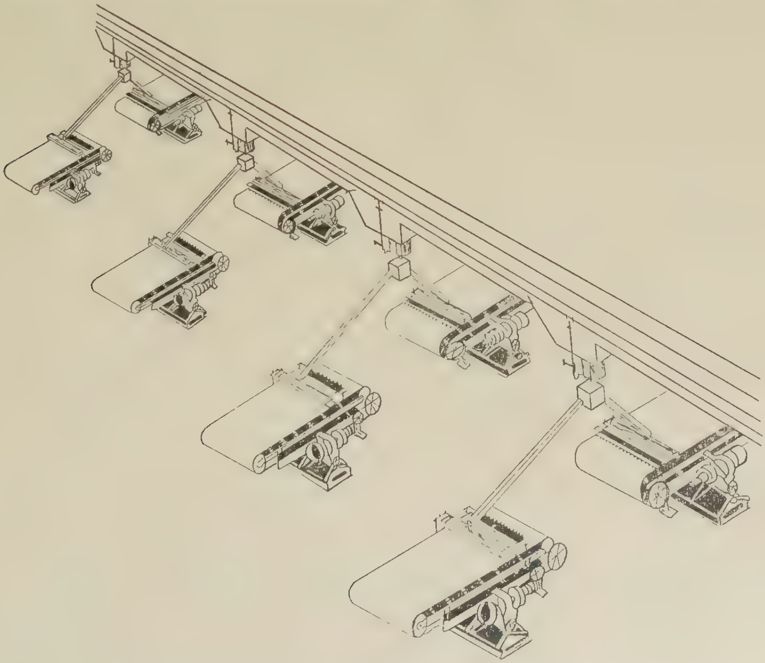

FIG. 197.

Launder Classifier.—General View of Installation. Richards pulsator-classifiers serving Isbell vanners (Allis Chalmers) (p. 285).

the surface pockets. By both arrangements the advantage is obtained that less fresh water is used than had all the classifiers been of the hydraulic type. It is the introduction of this fresh water which is the unavoidable disadvantage as well as the characteristic feature of these classifiers. On the coarse side it limits the size of material by reason of the great quantity of water necessary to obtain the desired uprising velocity; accordingly, hydraulic classification, though coarse when compared with that accomplished by surface classifiers, is rarely practised upon material as large as, say, $\frac{1}{4}$-in., though material crushed through $\frac{3}{8}$-in. has exceptionally been submitted. Depending upon the size of material treated and somewhat upon the number of classes made, the actual amount of fresh water used varies, say, from 3 to 6 tons per ton

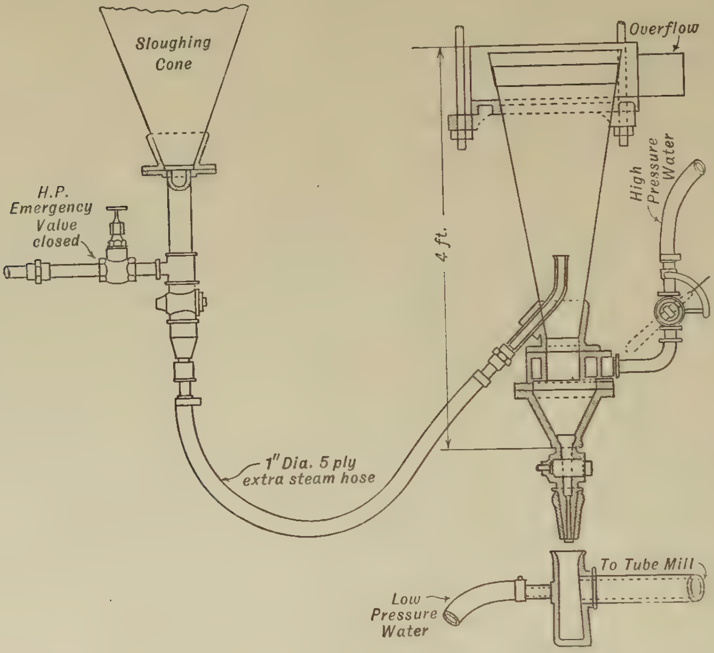

FIG. 198.

Cone Classifiers.—Sloughing cone and hydraulic cone in series. The underflow of the former becomes the feed for the latter, this feed being entered through a pipe directed upwards, so that the fine material continuing its rise to the overflow does not change its direction of movement. The pressure box and the short sorting tube are clearly indicated, as also are the hutch below the sorting tube, and the eventual spigot discharge. Apparently the resultant underflow is so thick as to make the provision of supplementary water advisable (p. 285, Fig. 173). (Clark and Sharwood, *Trans. I.M.M.* Vol. XXII., 1913, p. 92.)

of material entering the classifier, an amount not much less than that which enters with the material.

In a series, by far the greatest part of this water enters the coarsest classifier, whence also, as a rule, the greatest weight of product is discharged. The successively finer classifiers take progressively less water and deliver progressively less weighty products, leaving an overflow product which generally still contains a substantial proportion of the original feed. With the capacity determined by the relatively very small area of the sorting passage, hydraulic classifiers are not employed with material smaller than, say, 80 mesh, surface classifiers taking their place.

Hindered-settling Classifiers.—The complete hydraulic classifier may itself be regarded as a combination of a surface and a hydraulic classifier, the pocket, representing the former, relieving the sorting tube, representing the latter, of some of the work which otherwise would come upon it, thus reducing the amount of fresh water necessary. In turn, greater effectiveness of the sorting passage may be obtained by making it smaller and introducing above it an additional chamber in which the material to be sorted collects, to be so overhauled by the rising current that small and light particles are discovered and lifted, only those of proper size or density finding their way down the sorting tube proper

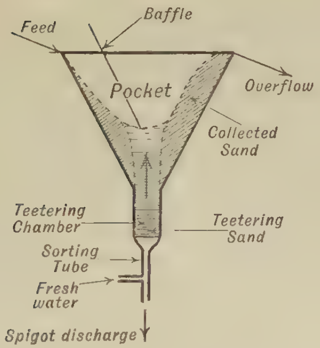

FIG. 199.

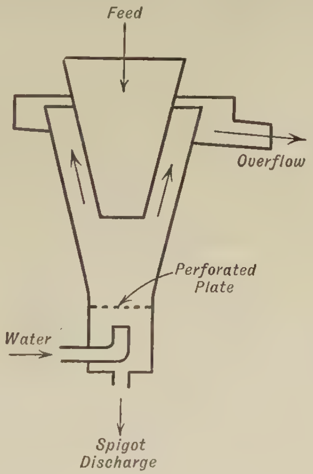

FIG. 200.

Hindered-settling Classifier (Fig. 199).—Diagram illustrating constant conditions of working (p. 287).

Hindered-settling Classifier (Fig. 200).—Diagram of the Anaconda Classifier. Hindrance formed by a perforated plate, the perforations being the sorting orifices. Above this plate comes the teetering chamber (p. 287).

(Figs. 199, 200, 201). This overhaul being effected by the water rising through the narrow interstices between the collected particles, demands a much smaller amount of water than were there no hindrance in the direct line of fall; in fact, the amount of fresh water required in these "hindered-settling classifiers," as they were termed by Richards, is little more than one-half that which would be required in free-settling classifiers.

Hindered-settling classifiers are accordingly those wherein the fresh water rises through the sorting tube or orifice into a chamber so dimensioned that the material collected there is brought into a quicksand suspension or 'teetering' movement. The area of the teetering chamber should not be greater than about four times that of the sorting passage or some of the material will lie quiescent; nor much less, because then the conditions of the ordinary free-settling classifier would be approached. With these conditions fulfilled and where particles of varying density and size are present, another specific advantage of hindered-settling is realized, namely, the small dense particle falls with much larger less-dense particles than under conditions of free fall; with quartz and galena, for instance, the diameter ratio of equal-falling particles under hindered-settling conditions is about 6 : 1—as though the specific gravity of the medium in which fall took place were about 1·5—instead of 4 : 1 which obtains in free-falling conditions. Of such an increased difference

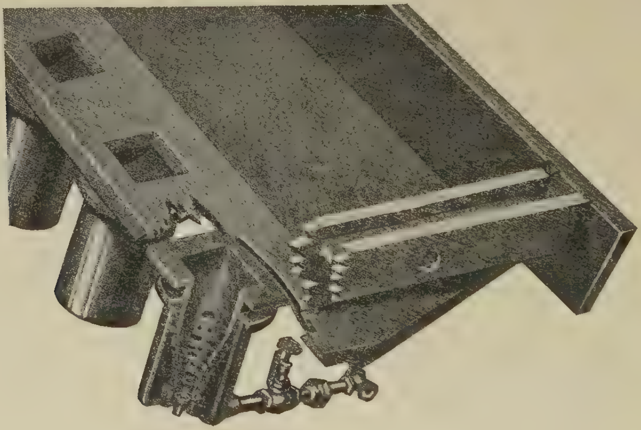

Fig. 201.

Tank Classifier.—View of Overflow End. (Richards hindered-settling classifier.) In this illustration the sorting tube of the last spigot is laid open to show the pressure box into which the water is delivered and the passages from this box into the teetering chamber. This latter is of expanding section upwards, its sloping sides constituting the obstruction to free settlement. At the bottom of the teetering chamber is the narrow passage which constitutes the sorting tube; and the discharge (pp. 287, 289).

in particle-diameter, advantage may be taken in the processes of separation which follow.

When such hindered-settling classifiers are in series, it is usual to forgo the advantages of separate pockets, and for convenience to place them all in the sloping bottom of a shallow tank or launder along which the material progresses, the particles being tested at the opening of each successive sorting tube. In such a series the openings should be rectangular and extend across the launder, or otherwise some of the particles will pass the opening down which they should properly descend (Fig. 201).

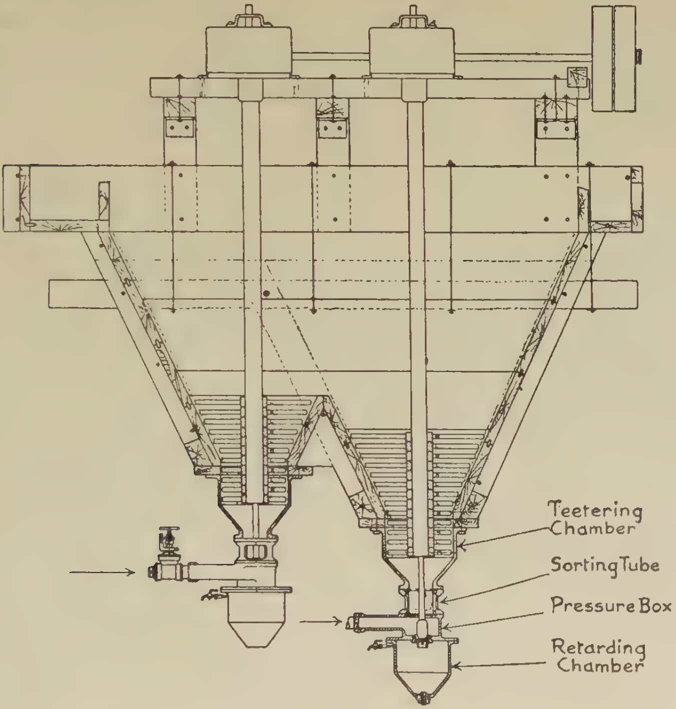

FIG. 202.

Richards-Janney Classifier. Cross-section (Allis Chalmers).—This classifier was designed for the large capacities associated with the disseminated copper mines of Utah and Arizona. A two-pocket classifier is shown, but five or six pockets are more common (p. 289).

In one particular design, however, the Richards-Janney classifier, each sorting tube is at the bottom of its own pocket, while each succeeding pocket is larger and stepped down from that preceding (Fig. 202). No forward progression is then possible by streaming or launder action. but banks and choking are prevented by the rotation of paddles extending down into the teetering chamber of each classifier, the rising current, taking advantage of the chance thus afforded, carrying any fine material upward. This classifier has the further distinction of an intermittent discharge—through a valve operated from above—an arrangement which permits a larger discharge-opening with consequently less risk of choking, and which saves water, because, discharge being discontinuous, in the intervals the material has time to collect and less water passes out.

A similar but more pronounced pulsating-effect is the distinctive feature of a Pulsator classifier designed by Richards, where, in the place of a continuous rising-current, a quickly-intermittent or pulsating current is used (Fig. 203). Such an arrangement, in allowing the sorted material to fall between successive pulses, and in necessitating the remaking of the rising current, prevents the establishment of disturbing eddies or irregularities, so that the quantity of uprising water is reduced to about 1—2 tons per ton of material.

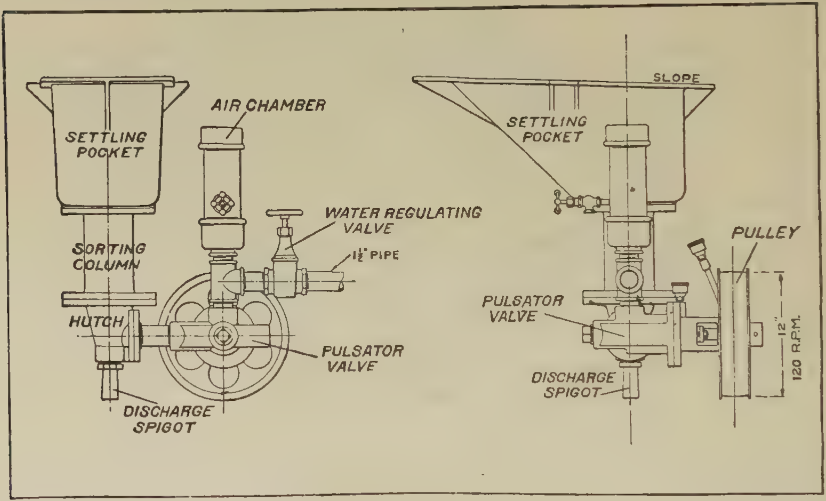

Fig. 203.

Richards Pulsator-Classifier.—Elevations (p. 290).

Water-sizing Efficiency

The results achieved by classification are best disclosed by screening analyses. Where, in the absence of minerals of different density, classification has been primarily a sizing operation, screening analysis of the products should show each to contain a decided preponderance of a particular size, with a rapid drop on either side. Poor classification is indicated when no very decided maximum appears, when the coarser spigot contains fine material, and when the coarse material is not

Screens.		Spigots.			Overflow.
		1	2	3	
– 20	+ 30	17	. .	. .	. .
– 30	+ 50	42	6	. .	. .
– 50	+ 80	22	18	8	. .
– 80	+120	10	36	14	1
–120	+200	5	21	23	4
. .	–200	4	19	55	95
. .	. .	100	100	100	100

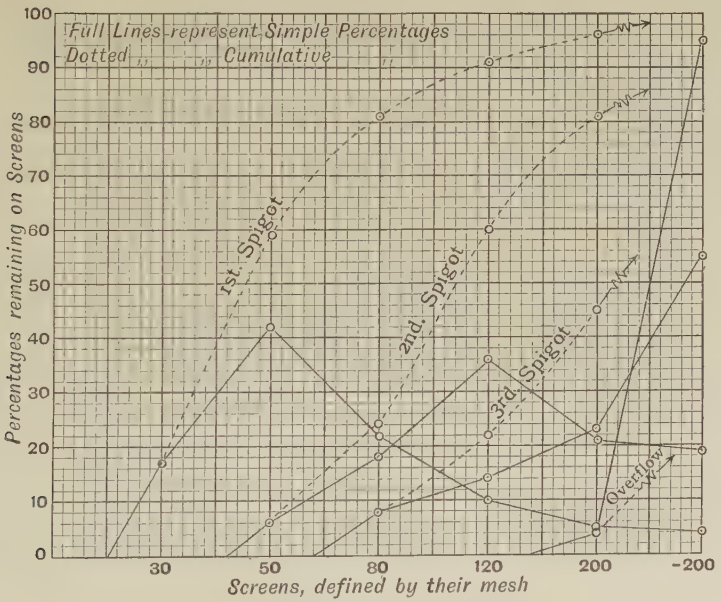

FIG. 204.

Classification Records.—Sizing analyses and graphs. The record given represents good work for a classifier. It is seen that no definite dividing lines are drawn by classification, yet each product is successively finer and each contains one particular size in marked amount; in the graphs of simple percentages (full lines) these particular sizes are the peaks. The more upright the graphs of the cumulative percentages (dotted lines) the better the classification; a flat line cutting the ordinates of many sizes represents poor work (p. 292).

exclusively confined to the coarse spigots. Where, however, mineral and gangue are both present, two maximum points might occur and the classification yet remain efficient. The best presentation of results is by means of graphs, an inspection of which will indicate the efficient or inefficient nature of the work being done (Fig. 204). Properly to appreciate the work done by a classifier-series, the sizing analysis of each product must be considered in relation to the weight of that product; as already stated, the coarsest product is generally the most weighty.

Analytically, from the point of view of sizing, the efficiency of a classifying operation is the degree of completeness of separation of the material larger than a predetermined size from the material smaller. Unlike a screening operation, however, not only may undersize be found in the coarse product but oversize in the fine product, and the efficiency suffers doubly.

The following reasoning and formulæ for this efficiency were developed by R. T. Hancock.[1] When the operation results in the complete separation of the oversize into the coarse product and of the undersize into the fine product, the efficiency is 100 per cent. Intermediate results are gauged by calculating the amount of unaltered feed in the coarse product—this being done by 'loading' the undersize in that coarse product with the amount of oversize necessary to give a product of the composition of the feed—then deducting the amount of oversize in that unaltered feed from that actually present in the coarse product; the remainder divided by total oversize in the feed is a figure expressing the efficiency of the operation.

Thus in separating from a weight of feed a, made up of ax of oversize and $a(1-x)$ of undersize, a Coarse Product of weight b, made up of oversize by and undersize $b(1-y)$—it will be realized that x and y are the sizing fractions of oversize in feed and coarse product respectively—the efficiency of the sizing operation is

$$\frac{by - \left\{(b-by)\frac{1}{1-x}\right\}x}{ax} = \frac{by}{ax} - \frac{(b-by)\frac{1}{1-x}}{a}$$

or, dividing top and bottom of the second term by $\frac{1}{1-x}$, this being the ratio of the weight of feed to the weight of undersize in the feed,

$$= \frac{by}{ax} - \frac{b(1-y)}{a(1-x)}$$

$$= \frac{\textit{Wt. of oversize in C.P.}}{\textit{Wt. of oversize in feed}} - \frac{\textit{Wt. of undersize in C.P.}}{\textit{Wt. of undersize in feed}}$$

= *Bulk fraction of oversize in C.P.* − *Bulk fraction of undersize in C.P.*

or, multiplying by 100 to obtain the percentage figures more commonly employed

= *Bulk percentage of oversize in C.P.* − *Bulk percentage of undersize in C.P.*

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[1] *E. and M. J.*, Sept. 25, 1920, p. 622.

or, the second term being equal to (100 – *Bulk percentage of undersize in F.P.*)

= *Bulk percentage of oversize in C.P.* – (100 – *Bulk percentage of undersize in F.P.*)

= *Bulk percentage of oversize in C.P.* + *Bulk percentage of undersize in F.P.* – 100

which is the final expression. The same result would have been reached if the considerations had been centred around the Fine Product.

It will be realized that applying this expression to screen sizing, where the whole of the oversize, 100 per cent, must of necessity be in the coarse product, the efficiency of screen sizing, as was stated when discussing that efficiency, is represented by the bulk percentage of undersize in the fine product.

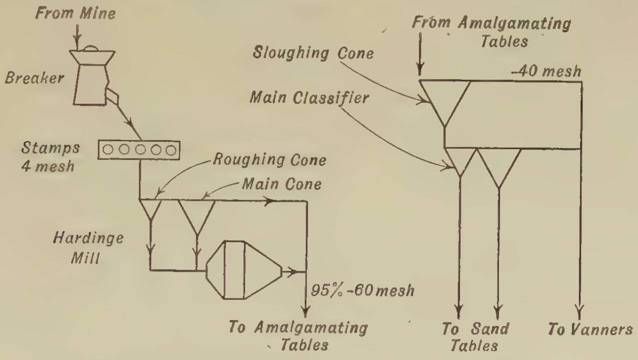

FIG. 205.

Classifying System at the Plymouth Mill, California (1914).—Auriferous quartz in slate country; about 2 per cent of pyrite and arsenopyrite in the ore crushed. Quartz friable and slate pulverulent, a good deal of impalpable slime being formed. Two-step classification was introduced in consequence of the buoyant nature of the resultant pulp. The roughing cone ahead of the Hardinge mill is a diaphragm cone; it separates most of the coarse sand, leaving the finer sand to be separated in the main cone. From the amalgamating tables the pulp passes into a sloughing cone which separates most of the slime, leaving less for the hydraulic main-classifier to separate. In 1916 this system of classification was changed to the extent that the spigot discharge from the sloughing cone was returned to the Hardinge mill for further grinding (p. 293, Fig. 173).

Finally, classification, as a preparatory operation dividing crushed material into classes for separate treatment subsequently, takes up the work where wet screening fails. The extent of such preparation depends upon the nature of the subsequent treatment. For the separation of sand for regrinding, classification performs a very definite and important service (Fig. 205). For the division into classes suitable for water-concentration, classification renders a service which, though equally definite, is one which of recent years has diminished in importance

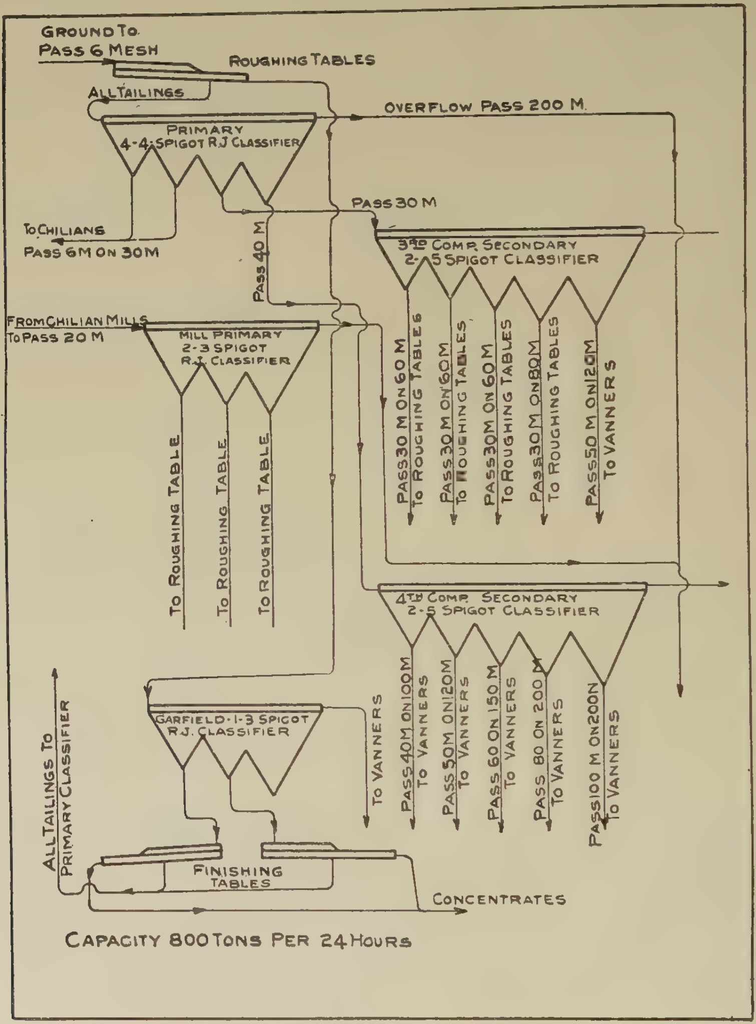
Ground To Pass 6 Mesh
Roughing Tables
All Tailings
Overflow Pass 200 M.
Primary 4-4 Spigot R.J Classifier
Pass 30 M
To Chilians Pass 6 M on 30 M
Pass 40 M
3rd Comp. Secondary 2-5 Spigot Classifier
From Chilian Mills To Pass 20 M
Mill Primary 2-3 Spigot R.J. Classifier
Pass 30 M on 60 M To Roughing Tables
Pass 30 M on 60 M To Roughing Tables
Pass 30 M on 60 M To Roughing Tables
Pass 30 M on 80 M To Roughing Tables
Pass 50 M on 120 M To Vanners
To Roughing Table
To Roughing Table
To Roughing Table
4th Comp. Secondary 2-5 Spigot Classifier
All Tailings To Primary Classifier
Garfield 1-3 Spigot R.J. Classifier
To Vanners
Pass 40 M on 100 M To Vanners
Pass 50 M on 120 M To Vanners
Pass 60 on 150 M To Vanners
Pass 80 on 200 M To Vanners
Pass 100 M on 200 M To Vanners
Finishing Tables
Concentrates
Capacity 800 Tons Per 24 Hours

Fig. 206.

Classifying System at the Utah Copper (1912).—This flow-sheet represents the classification of the sandy material of a disseminated copper-ore for water concentration. Though a high development of classification is shown, the fact that all the material passing 6 mesh first proceeds without division to 'roughing' tables indicates a curtailment of the task allotted to classification. 'Finishing' tables and vanners eventually produce the final concentrate. With the advent of flotation the curtailment of classification was continued almost to extinction (pp. 295, 646).

because of the employment of water-concentrating machines themselves in a "roughing" or preparatory capacity (Fig. 206); doing such work, water-concentrating tables do not, like classifiers, further dilute the slime suspension, but the fresh water added passes away with the sand tailing, from which it is readily separable. With flotation concentration the machine itself is capable of handling the crushed material without division, and further classification becomes unnecessary.

In water concentration, to remove slime from sand before sending the latter to the sand table, is often described as "de-sliming," and the classifier in which this removal is effected, as a de-slimer. On the other hand, a "de-watering" device is generally regarded as one which separates water from a granular material, tailing or concentrate, the term "thickening" being reserved for the separation of clear water from thickened or settled slime.

Though classification like screening is usually a preparatory operation, like screening it may sometimes give final products either of value or for discard, a fact mentioned when introducing the subject of sizing. For instance, the necessary appliances being cheap to install and to operate, it occasionally is applied to the production of an enriched spigot discharge from material so poor as to preclude any more expensive and better treatment, such as that by concentrating-tables direct. Flotation tailing has in this way sometimes provided material for direct treatment on tables (Fig. 329). It has also been proposed by classification to recover clear pyrite from the pyritic material often associated with coal seams, the price of pyrite not warranting the installation of more expensive apparatus. Generally, however, the enrichment obtained by classifiers is insufficient, and, at the same time, much fine mineral escapes; in consequence, the final operation is best performed by some other appliance.

Position at the End of the Preparatory Operations

Crushing and a division of the crushed product into classes, result in the following materials:

Mixed Material, which has not been divided, or divided only between wide limits;

Sized Material, which has been divided into classes according to size, each class containing mineral and gangue of equal size; and

Classified Material, which has been divided into classes according to both size and density, each class consisting of small mineral-particles and relatively large gangue-particles. From the coarsest of a given series of such classes, it might very well be that the gangue had been completely driven. The finest class, on the other hand, would always contain more than a normal proportion of gangue (Fig. 207).

CHAPTER VIII

WATER CONCENTRATION

General Principles.—The material submitted to water concentration has been prepared by crushing and by division into classes (Fig. 207). If, as result, it be 'sized' material, then, from each class, size as a variable will have been eliminated, and density would be capable of coming freely into play. The submission of such sized-material to the discriminating resistance of water, as in vertical rise and fall, would result in the separation of the mineral, in which event the double sequence of a preparatory sizing followed by classification, would have been effective. Or, presupposing a fine condition, if the material had been prepared by water-sizing, each class would consist of fine particles of mineral and relatively coarse particles of gangue, and a ready separation could be made by diametral sizing. The diametral sizing of such fine material being, however, beyond the economic possibility of ordinary screening, the necessary sizing is effected by horizontally-moving films of water, that is, by "film-sizing," friction with the surface

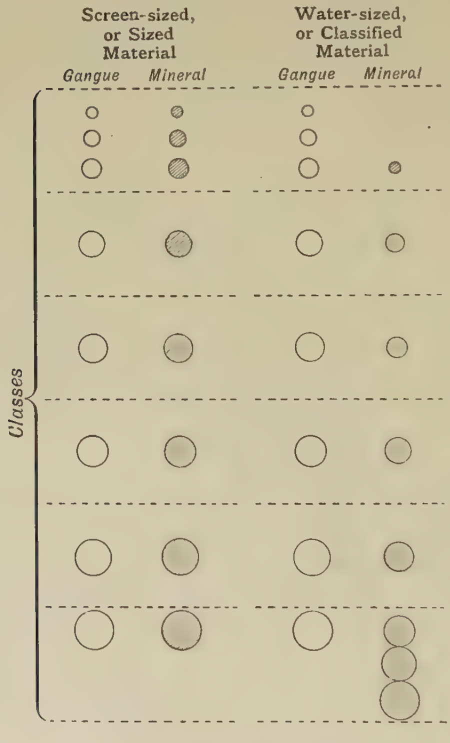
Screen-sized, or Sized Material
Gangue
Mineral
Water-sized, or Classified Material
Gangue
Mineral
Classes

FIG. 207.

Sized and Classified Materials.—Diagram. It is usual to speak of screen-sized products as 'sized' material, and of water-sized products as 'classified' material. The finest classified product, that is, the final overflow, contains a disproportionate amount of gangue, the coarsest product an equally disproportionate amount of mineral (pp. 196, 296).

over which flow takes place being the discriminating resistance (p. 257). The double sequence with 'classified' material will accordingly be:

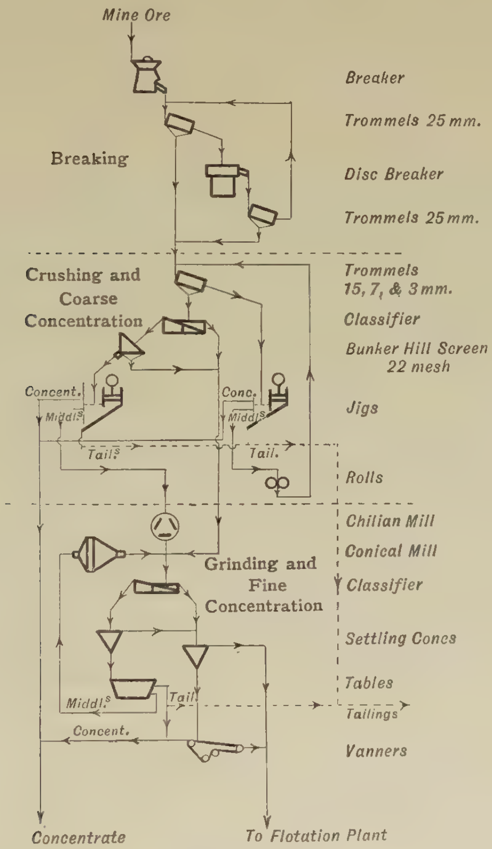

Fig. 208.

Scheme of Comminution, Sizing, and Water Concentration at the Bunker Hill Mine, Idaho.—Flow-sheet. The ore at this mine contains about 10 per cent of argentiferous lead, the ore-mineral being galena, and the gangue containing a good deal of siderite and pyrite, in addition to lighter material. This flow-sheet shows very well the relation of breaking, crushing, and grinding, as well as that of coarse concentration, fine concentration, and slime concentration, this last being represented by flotation. The sizing appliances, trommels, rotating screens, drag classifiers, and finally spitzkasten, are also shown in their place (pp. 297, 378, 646).

classification followed by film-sizing, that is to say, a sequence the reverse of that previously quoted.

Material which allows itself to be screened is relatively coarse, and the operations for its beneficiation constitute Coarse Concentration (Fig. 208);

such operations are characterised by vertical movement of the water. Classified material, on the other hand, connotes Fine Concentration, the operations of which are characterized by flowing or roughly horizontal movement. Beyond that again, the endeavour to concentrate the very finest material is Slime Concentration, this being also conducted horizontally.

COARSE CONCENTRATION: VERTICAL–CURRENT CONCENTRATORS

JIGS

In the coarse concentration of crushed material, practically the only appliance is the jig, a machine wherein the material is submitted to vertical movements of water (Figs. 222, 223, 225-227). The design and operation of this appliance are well illustrated by the hand jig.

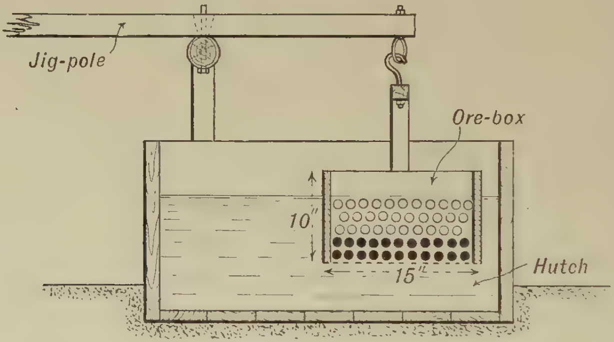

FIG. 209.

Hand Jig.—Diagrammatic Cross-section. The diagram shows mineral particles below and gangue particles above (p. 298).

Hand Jig.—This jig consists of a tank or hutch containing water, in which an ore-box with a sieve for bottom is moved up and down (Fig. 209). The ore-box is, say, 15 in. wide by 24 in. long and 12 in. deep, though exceptionally, as at Joplin, its dimensions may be as much as 24 in. by 60 in. by 10 in. Having uprights at each end and a cross-piece joining them, this box is attached to a pole by means of an eye-bolt which loosely engages a hook passing through the pole. The pole itself is 15—20 ft. long, the greater portion of this length being on the other side of a support over which the pole rocks. The operator, standing at the far end of the pole, raises and lowers the box at a rate and with a movement suitable to the material being jigged. In detail, this movement consists of a quick downward movement into the water, followed, after a short pause, by a slower upward return. The amplitude of this movement is generally somewhat more than an inch, and the frequency about 80 strokes per minute. The material submitted has generally been hand-broken to about an inch-ring and closely sized, the finest material being laid aside as not permitting a return commensurate with the effort demanded. For material of an inch or so, the sieve at the bottom of the ore-box would have an aperture of about $\frac{1}{2}$ in.; for half-inch material, an aperture of about $\frac{1}{8}$ in., and so on.

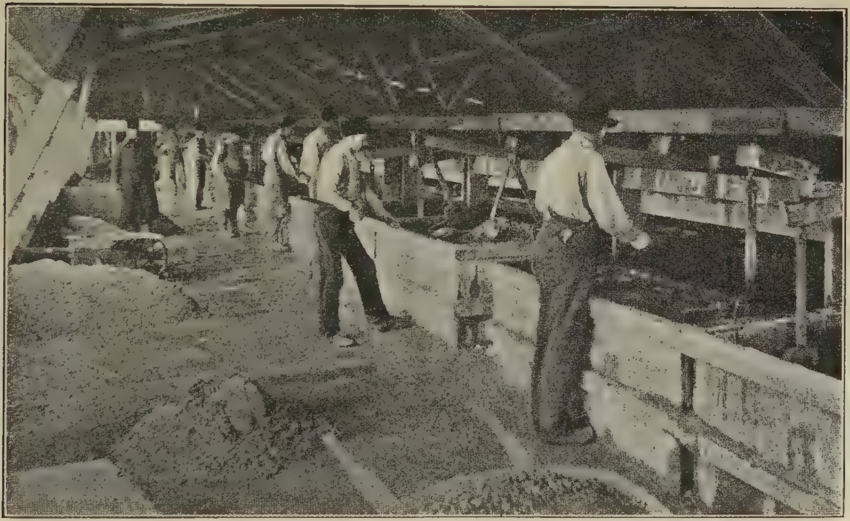

Fig. 210.

Hand Jig.—Photograph (*M. & S.P.*, Feb. 1, 1919). The front or free side of the hutch is shown. This jig at Joplin is so large, 2 feet by 5 feet, that it takes one operator to attend to the ore-box, while another, who is out of the picture, works the jig pole. An imposing equipment of these hand jigs is shown (p. 300).

The procedure is as follows: a charge having been placed in the box, the jig is worked until stratification has brought the gangue to the top, the mineral having forced its way below. The gangue is then scraped off. In its place a further amount of ore, or "feed," is added, and the operation repeated till the collected mineral so disturbs the balance of the machine that it is no longer easy to continue. The clean mineral is then taken out and the work continued on an entirely fresh charge.

It is seen that the operation produces clean mineral and impoverished gangue; the former is described as the "concentrate" and the latter as the "tailing." In addition to these two products, an amount of fine material, present partly by reason of imperfect sizing, partly as the result of attrition arising in the movement, passes through the sieve and collects in the hutch; this third product is known as "hutch-work." Perhaps, also, after scraping off the tailing, it has been possible to remove an intermediate layer which was neither clean mineral nor poor tailing, and to set it aside for further treatment; such a product would be described as a "middling."

Doing the work described, the hand jig of small size might treat one and a half tons per shift, and that of large size 10—15 tons. To render these figures of capacity comparable with those obtained with mechanical jigs, described later, they are best converted into tons per square foot of sieve area per day of 24 hours, in which terms hand jigs have a capacity of about 2 tons.

Though the capacity is limited, hand jigging is very useful in the early stages of a mine's existence, useful also in beneficiating small and uncertain ore-bodies, and in testing development rock. It needs very little water, the only losses of water being that due to splashing and that carried away as moisture in the products. It is, however, only applicable to simple ores, since it makes only a one-mineral separation, that is to say, it separates only one mineral at a time. Hand jigging is at present in considerable use in the Joplin district, Missouri, for zinc; in Bolivia for wolframite and tin; and at other scattered places (Fig. 210).

The Water Action in Jigging.—When the ore-box descends, the water rushing through the sieve lifts the bed of material, forcing the particles each to its position under the hindered circumstance of a crowded assembly. This upward movement of the water is the "pulsion stroke." In response to it, if the material has been suitably sized, that is, if the ratio of the diameter of the largest particle of gangue to that of the smallest particle of mineral is within that between the respective densities each diminished by 1·5—the effect of the hindered circumstance being virtually to increase the density of the medium in which the particles move, making it 1·5 instead of unity—and if the stroke has been properly proportioned, all the gangue particles, even the largest of them, will be above the mineral particles, even the smallest (Fig. 211).

Immediately following, or after a slight pause, comes the downward or "suction stroke" of the water as the ore-box is raised, the particles then falling in a descending current. In such a current the discriminating resistance of the medium being weakened, or, it may be, reversed, there will be a tendency—not necessarily an overriding tendency—to carry the gangue rapidly downwards and thereby to undo the good work done on the pulsion stroke; hence the slight pause to allow fall to begin in still water. Accordingly, with closely-sized material, the suction stroke is made slowly, the pulsion stroke, on the other hand, smartly. A further disadvantage of strong suction with such material, is that it compacts the bed, whereas for quick separation a loose bed wherein the particles may quickly take up their proper positions, is desirable.

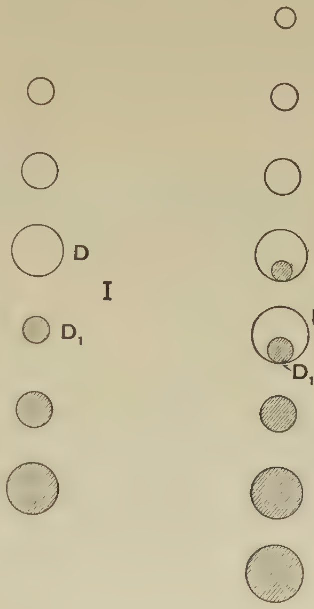

FIG. 211.

Jigging.—Diagram illustrating the effect of the pulsion stroke.

I. When the size-relation between the smallest particle of mineral of diameter D, and the largest diameter of the gangue D has been within that demanded by hindered conditions, that is, when

$$\frac{D}{D_1} < \frac{\delta_1 - 1{\cdot}5}{\delta - 1{\cdot}5}.$$

II. When such close sizing has been exceeded, that is, when

$$\frac{D}{D_1} > \frac{\delta_1 - 1{\cdot}5}{\delta - 1{\cdot}5}.$$

The small particles of mineral shown up among the gangue particles would be such as could subsequently be pulled down by strong suction (p. 300).

Supposing, however, the sizing had not been close but had largely exceeded the ratio mentioned, a strong suction stroke would draw fine particles of mineral down through the interstices between the larger particles, before these had locked themselves to bar the passage; the equal-falling non-metalliferous associates of these fine particles would, by their greater size and late arrival, be prevented from making the same downward movement. In this way a true and useful sizing effect results from the suction stroke. It should be remarked, however, that if it be considered that, at the end of the pulsion stroke, fall begins in still water, the dense particle, whatever its small size, would drop quicker than the less dense particle, since, at the momentary beginning of fall, density alone counts (Fig. 163).

It is obvious from the above considerations that by proper manipulation of the suction stroke the possibilities of jigging may be substantially increased, and that strong suction or weak suction may make all the difference. Of these possibilities use is made in mechanical jigging. "Strong suction" obtains when all the water which passes up through the bed during the pulsion stroke—or an equivalent amount of water—passes down through the bed again on the suction stroke. This position can be conceived by assuming the sides of the ore-box to project so far above the water in the hutch, that none of the water passing up through the bed can overflow into the hutch. Starting to jig under such conditions the particles on the bed would behave as valves, opening to let the water pass upwards during the pulsion stroke, but closing to hinder the return movement downward. This would continue until the water pumped in this way above the bed, reached such a level that under its pressure the leakage downwards through the bed was equal to the amount passing upwards.

The condition of "mild suction," on the other hand, obtains when a portion of the water, say one-half that which passes up through the bed on the pulsion stroke, overflows into the hutch, leaving but the remaining half to descend. This condition keeps the bed loose and favourable for jigging. It is that chiefly used in jigging sized-material.

Finally, when none of the water which passes upwards during the pulsion stroke returns during the suction stroke, the condition of "pulsation" without suction obtains, a condition reached in the Pulsator Classifier, in which, as already described, intermittent pulses of water pass upward, the water being at rest during the intervening intervals.

MECHANICAL JIGS

Mechanical jigs may be divided into: plunger jigs, with a fixed sieve, such as the Harz jig and the Joplin jig; tray jigs, with a moving sieve, such as the Hancock jig; pulsator jigs with water impulses, such as the Richards jig; and paddle jigs, such as the Neill jig.

The Plunger-Jig Cell.—In this cell an ore-box, of width 18 in. to 24 in., of length 24 in. to 48 in., and of depth 6 in. to 8 in., is fixed in one longitudinal half or compartment of a hutch, in the other half of which a plunger moves up and down (Fig. 212). These two compartments are separated by a centre-board which continues below the sieve to a depth equal to about half the sieve width, sufficient to secure an even arrival of the water impulses at the sieve. The bottom of this hutch is generally inclined from the plunger compartment to the jigging compartment, to facilitate the collection and discharge of the hutch-work in front; below the centre-board the water passage must be sufficient to pass freely all the water set in movement, that is to say, it should have an area not less than that of the plunger. A pyramidal bottom is sometimes seen, and, more rarely, a semicircular bottom.

The plunger itself is generally rectangular in shape and of similar area to the ore-box. It fits loosely in its compartment, this looseness minimizing friction, wear, and water-shock; and receives its movement from an eccentric upon an overhead shaft. When this eccentric is of normal design, as it usually is, it does not differentiate between the pulsion and the suction strokes (Fig. 213), but when of differential design, a useful difference between these two strokes is made (Fig. 214). Or, its harmonic motion may be conveyed to the plunger through differential lever-mechanism (Fig. 215). Usually, however, any difference between the strokes is obtained by water adjustment, differential mechanisms having the disadvantage of complication.

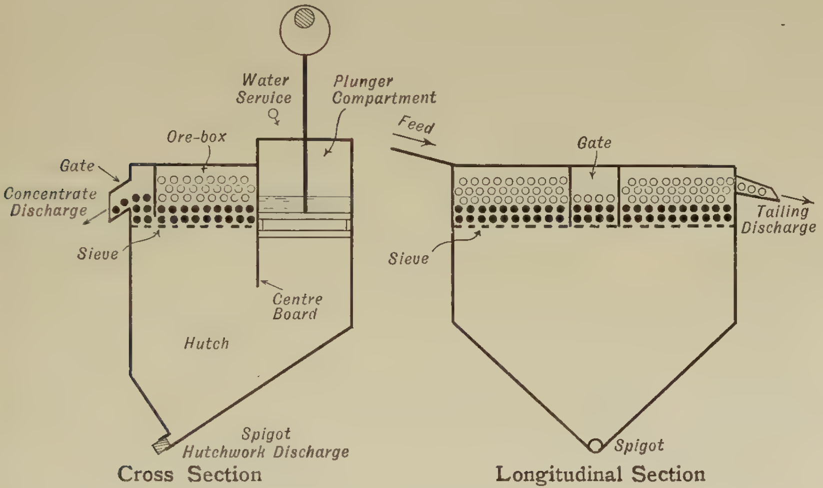

Fig. 212.

Plunger Jig.—Diagrammatic Sections of a Single Cell. These sections illustrate a gate discharge for the concentrate, and the overflow discharge of the tailing (p. 302).

The sieve obviously plays an important part in the proper functioning of the jig. For small sizes it is of woven wire, for middle sizes of punched sheet, for larger sizes of barred grates. Woven wire, being weak, is held between an upper and a lower grate to prevent sagging, being attached to

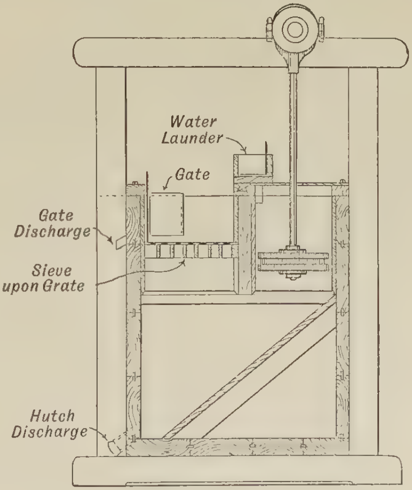

Fig. 213.

Plunger-Jig Cell; Normal Design.—Cross-section. Plunger actuated by an ordinary eccentric. Gate discharge, hutch discharge, and clear-water launder are shown. Simple inclined bottom (p. 303).

the latter; both these grates have their bars widely spaced, so as not to restrict the water-way (Figs. 213, 214). Punched sieves are generally of stiff plate, needing only a grate beneath; the holes are generally round, if slotted, the length of the slots is placed across the stream; steel is generally the material employed, but, where the water is acid or contains copper in solution the plate is of copper or brass. Barred grates are generally of cast iron, and need no extra grate in support; they are placed with the bars across the stream; with them the water-way is less than with punched plate of the same aperture, but, on the other hand, they are cheaper and more easily cleaned. Whatever the sieve, there is a tendency for it to become blinded by particles which have become pulled down and fixed in the apertures by the suction stroke. Sieves, accordingly, require regular cleaning; barred grates are easiest cleaned, wire sieves are cleaned with difficulty.

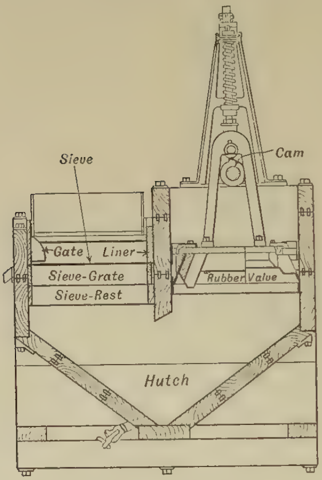

Fig. 214.

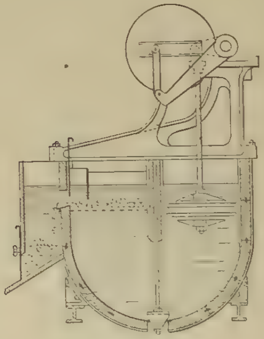

Fig. 215.

Plunger-Jig Cell; Pulsation Stroke.—Cross-section (Fig. 214). The upward movement of the plunger is made by a cam working against a spring. During this movement, valves on the sides of the plunger open to let the plunger slip through the water, to which consequently no suction stroke is conveyed. The pulsion stroke, on the other hand, is made smartly by the spring, the valves closing immediately. The sieve here sits upon a 'grate,' and this in turn upon a 'rest.' The gate is seen to be a hooded shield leading to the discharge. The hutch discharge is through a 'molasses' spigot (p. 303).

Plunger-Jig Cell; Differential Stroke.—Cross-section (Fig. 215). Difference between the two strokes obtained by levers and links. Gate-discharge into a stay-box to diminish the amount of water discharged with the concentrate. Rounded bottom (pp. 303, 309, 313). (Commans, *Proc. Inst. Civil Eng.* Vol. CXVI., 1893–94.)

Above the sieve and generally in the middle of the free side of the ore-box, is a gate or cup through which concentrate may be discharged.

This gate consists of a semi-cylindrical shield guarding an otherwise open discharge, in such a manner that the only passage to the discharge is under the gate (Figs. 212–215). The concentrate collecting on the bed closes the passage to the gangue, the weight of which in the column outside the gate forces the concentrate to rise within the gate, discharge being within the height of this rise. The concentrate accordingly creeps under the gate, some being discharged at each pulsion stroke, while the gangue passes on with the stream to be discharged over a lip at the end. A similar gate is sometimes arranged over a central pipe, down through which the concentrate passes instead of being discharged at the side (Figs. 216, 217).

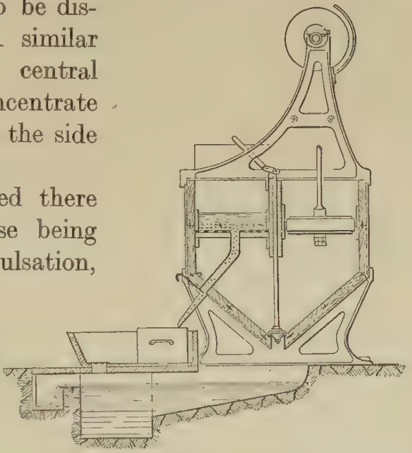

Fig. 216.

Plunger-jig Cell ; Pipe Discharge.—Cross-section. In this illustration, gate discharge is replaced by pipe discharge. The lower end of this pipe is only kept so much open that the amount of concentrate discharged is equal to that arriving, and the bed, in consequence, is maintained at constant depth. The opening of the hutch discharge is effected by a handle on top. Double-sloping bottom (p. 305). (Commans, *Proc. Inst. Civil Eng.* Vol. CXVI. 1893–94.)

From the standard cell described there are many variations, most of these being designed to secure a more even pulsation, connoting a more even bed and better work. In the Joust jig, for instance, considerably used at Joplin, there are two plungers, one on each side of a central ore-box of very large dimension, these plungers having a combined area somewhat greater than that of the sieve (Fig. 217). The Hodge jig, on the other hand, has a central plunger with an ore-box on either side, each of these ore-boxes having half the width of the plunger (Fig. 218). In the Parsons jig, formerly considerably used in the Missouri lead belt, a horizontal plunger works in a sleeve piercing the wall common to two jig cells in series, serving them both, and fitting its sleeve so closely that a sufficient pulsation is obtained with an area only one-third that of the ore-box (Fig. 219). In the Doubledee jig the plunger is in the same compartment as the ore-box, and below it, the rod by which it is worked passing up through the ore bed, a sleeve attached to the sieve keeping this bed intact; with such a close and direct association of plunger and sieve, a diminished stroke has been found sufficient, and a very even bed results (Fig. 220). In the plunger of the Overstrom jig are flap valves which, on its upward movement, open to allow the plunger to slip through the water without causing suction ; this jig, accordingly, works with a pulsion stroke only. With the Woodbury jig, the two compartments of the cell, plunger and ore-box, are not disposed across the flow but along it ; the plunger compartment is then at the head of the jig and the pulsation wave moves with the flow, suffering no such transverse deflection as conceivably happens with the normal plunger-cell (Figs. 221, 228).

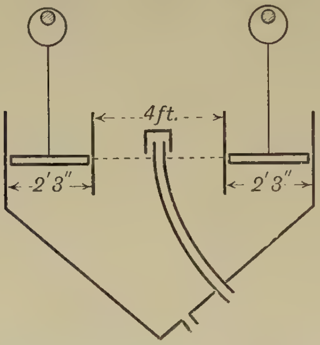

FIG. 217.—The Joust Jig (p. 305).

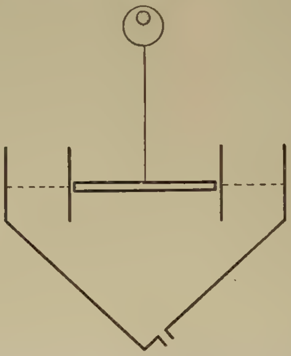

FIG. 218.—The Hodge Jig (p. 305).

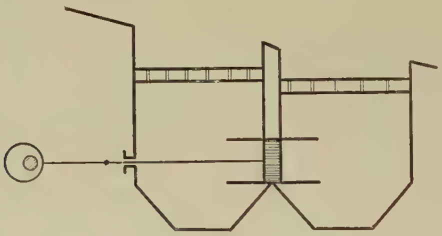

FIG. 219.—The Parsons Jig—Two cells in series (p. 305).

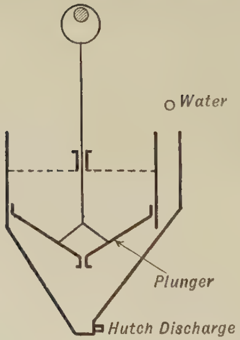

FIG. 220.—The Doubledee Jig (p. 305).

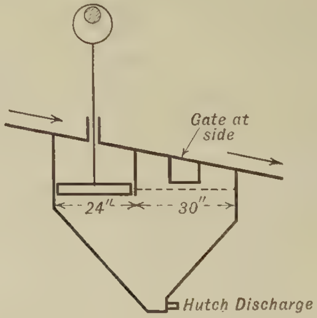

FIG. 221.—The Woodbury Jig (p. 306).

To make good the water which passes away with the products, and, when the material is not water-borne, to provide sufficient water to carry it forward, water has to be supplied; it is usually added in the plunger compartment. If, then, there be no discharge from the hutch, this added water must pass upward through the bed, reinforcing the pulsion, and working against the suction stroke. If, however, there be a continuous discharge of concentrate from the hutch, the added water may be only sufficient to effect that discharge, and the two strokes will be unaffected. Finally, where the hutch discharge takes with it more water than the water added, some of the water entering with the feed will have to come down through the bed, and the suction stroke will be reinforced.

Methods of Jigging.—The different effects obtainable by manipulating the suction stroke permit two methods of jigging, namely: "jigging over the sieve," when, with mild suction, the concentrate is formed upon the sieve; and "jigging through the sieve," when, with strong suction, the mineral is pulled down through an artificial bed of larger material, and then through the sieve into the hutch below. The first method, having originated in Germany, is known as the German method. The second, a modification introduced when finer material came to be treated, is known as the English method. With the German method three products inevitably result, a concentrate, a tailing, and the hutch-work; the sieve aperture being a little smaller than the smallest mineral present, the particular application of this method is with coarse material, or otherwise the sieve aperture would be too small for an adequate passage of water. With the English method there are, as a rule, only two products, the hutch concentrate—which of necessity includes the hutch-work, and is consequently not quite so clean—and the tailing; since the artificial bed constitutes the necessary bar to the descent of gangue, the sieve aperture may be, and is, larger than the mineral in the ore treated; this method, accordingly, is particularly applicable to fine material.

Originally, plunger jigs handled only coarse material, and the concentrate was formed and drawn off over the sieve, through the gate. Below the gate and upon the sieve, a sufficient depth of concentrate collects to form a natural bed, the office of which is to keep the gangue, and particularly the fine gangue, from getting into the hutch. The depth of this bed is generally about 2—3 in., being deeper for fine material, for poor material, and for complex material, it being otherwise not possible to obtain a clean concentrate from materials of these descriptions. Too deep a bed is, however, a mistake, since more power is consumed in raising it, and more mineral is rendered unsavable by the resulting attrition. With easy material, shallow beds are accordingly used. Even with closely-sized material, a certain amount of hutch-work is always formed, the fine material resulting from attrition and that present from imperfect sizing, creeping down in eddies and backwaters. The two main products, however, are a clean gate-concentrate and a clean tailing.

To-day, however, with the plunger jig the English method is also widely employed, especially with fine classified-material but also with sized material up to about 8 mm. Using this method, the sieve aperture is a shade larger than the coarsest particles of the feed, the gangue being prevented from passing these apertures by an artificial bed of material of greater density than the gangue and of greater size than the sieve aperture. There is obviously needless consumption of power if the bed material is of unnecessarily great density; usually it consists of larger pieces of the same mineral as that being saved, these larger pieces being generally taken from one or other of the coarser jigs. Generally this bed is about 2½ in. deep for coarse material, and 3½ in. for fine material, these depths being a little greater than with the German method of jigging, since with that method the grate itself serves as an additional barrier to the downward passage of the gangue. On the other hand, jigging through the sieve tends to keep the bed even, since discharge of the concentrate takes place over the whole area of the sieve, and not at one place only as with the gate discharge.

It being possible by this method of jigging to save fine mineral, the range of size possessed by the material undergoing treatment is many times greater than when employing the German method. With this wide range, which experience shows may be roughly five times the ratio of the densities diminished by 1·5, *i.e.* $5\left(\frac{\delta_1 - 1{\cdot}5}{\delta - 1{\cdot}5}\right)$, this method of jigging is particularly suited to classified material, treating which it gives a reasonably clean hutch-concentrate and a clean tailing.

Using the method proper under the particular circumstance, the plunger jig is capable of treating material as coarse as 2 in., while when treating material 1 mm. in average size it can save mineral as small as 120 mesh. The number of strokes and the amplitude of the stroke must be suitable to the particular size, the number varying from 100 to 300, and the amplitude from 2 in. to ⅛ in. Exceptionally, at Anaconda, Montana, where 2 in. material is jigged, the stroke is as much as 5 in. For 1 in. material the normal stroke is about 2 in.

The amount of added water participating in this work is generally about 2—5 tons per cell per ton of ore, the precise amount depending upon the method of jigging, and upon the amount of water entering with the feed. Coarse feed, for instance, is generally fed relatively dry, and to secure progression of the products to their respective discharges, water must be added in the jig itself. When jigging through the sieve, the spigot discharge at the bottom of the hutch is often kept open, though the actual passage is restricted; when only opened intermittently less water is consumed. When discharging the concentrate over the sieve the gate is always open, but discharge only takes place on the pulsion stroke, and the discharge lip is kept as high as possible. The same necessity to minimize the consumption of water places the lip of the tailing discharge at as high a level as is consistent with free discharge. Exceptionally, and more particularly with fine material, the discharge of the water is stayed in an attached box (Fig. 215).

COMPLETE JIGS AND JIGGING SYSTEMS

Despite their importance, the greatest variations in jig design and practice are not in the disposition of the organs which go to make a single cell, but in the assembly of the cells together in series to make a complete jig, and in the relation of such complete jigs to one another, this latter relation constituting what might be called "jigging system." Where all the mineral has been released by crushing, it is conceivable that within given size-limits complete separation could be effected in a single cell. That being rarely, if ever, the case, cells have to be assembled in series, the number thus put together depending upon the number and difficulty of the separations to be made. It is usual to describe these assemblies as 'jigs,' though generally it is the jigging system which is meant. So defined, two plunger jigs merit special description, viz. the Harz jig and the Joplin jig.

The Harz Jig.—With the Harz jig the material to be treated is prepared by close sizing, the coarser material by screening and the finer by classification (Fig. 222). With the coarser material the German method of jigging 'over the sieve' is employed, and with the finer material the English method of jigging 'through the sieve,' the dividing line being at about 8 mm. On the coarse side, material about 1½ in., and on the fine side, sandy material of about 50 mesh, are the extreme sizes. Exceptionally, material as large as 2 in. is jigged, at the Anaconda mine, for instance, while on the fine side, recoveries of mineral as fine as 120 mesh may be made when such mineral has passed into the jig with coarser material. The necessary preparation of the material consists in sizing it very closely, the sieve ratio throughout being generally less than 2. In that way many separate jigs go to make the complete system, this system often comprising coarse jigs, intermediate jigs, fine jigs, sand jigs, and middling jigs, each having a proper number of cells (Fig. 223). Accordingly, Harz jigging represents "class concentration" (p. 313).

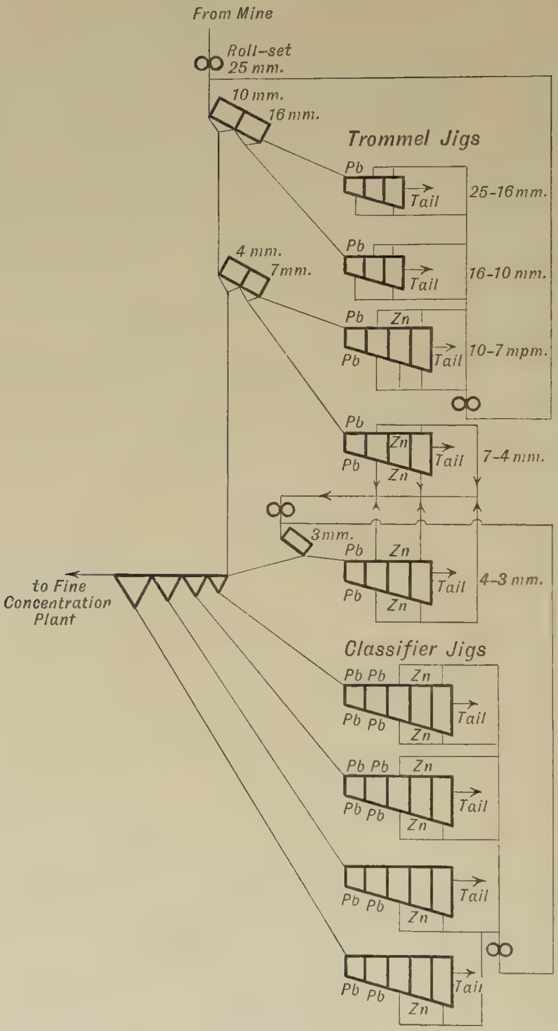

FIG. 222.

Harz Jigs at the Van Roi, Slocan, British Columbia.—Flow-sheet of Coarse Concentration Plant. Ore contains galena and blende, together with siderite, pyrite, quartz, and slate; it assays about 5 per cent of lead, 10 per cent of zinc, and 15 oz. of silver. The galena concentrate assays 65 to 70 per cent lead; the blende concentrate 45 per cent zinc. The percentage recovery of the whole plant, fine concentration included, is 80 to 85 per cent of the lead, 40 to 60 per cent of the zinc, and roughly 80 per cent of the silver.

The three coarse jigs make the cleanest tailing and are responsible for about 70 per cent of the total tailing. The classifier jigs and the remaining trommel-jigs account for 20 per cent of the tailing; and the fine-concentration plant (Wilfley Tables) for the remaining 10 per cent. It is interesting to note that all the four rolls are of the same size, 36″ × 12″ (pp. 242, 298, 309, 311, 378). (*E. & M.J.*, Mar. 11, 1916, p. 465.)

As already stated, the number of cells depends upon the number and difficulty of the separations to be made. In the simplest case, with only one mineral to be separated and the ore coarsely aggregated, it will be found that two cells in series are sufficient, a concentrate being recovered from the first, a middling from the second, this latter cell at the same time discarding the final tailing (Fig. 224); but, where the mineral is fine, three cells might be necessary even to make a one-mineral separation. With complex ores having two minerals to be separated, four cells are usually necessary, two to separate the respective minerals, and two for the middling products, leaving a tailing to be discarded. With fine complex material, five cells are common practice (Fig. 222).

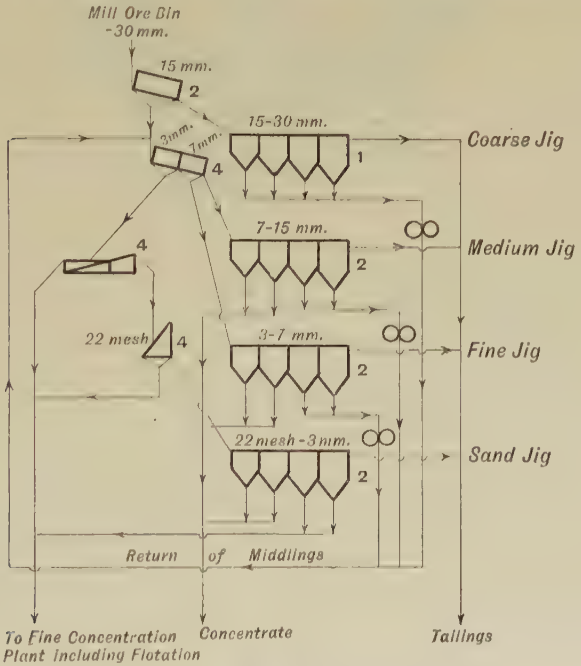

FIG. 223.

Coarse Concentration at Bunker Hill, Idaho.- Flow-sheet. A modified Harz System, four sizes of material being jigged. The coarse jig is a roughing jig separating a middling product to be recrushed, and discarding a final tailing. The other jigs all separate clean concentrate, a middling to be returned, and a final tailing. The block figures represent the numbers of the several machines in a unit plant treating 500 tons per day (pp. 298, 309).

Whatever the number of cells, each complete jig maintains a constant aperture throughout, and also a constant number of strokes. Each cell,

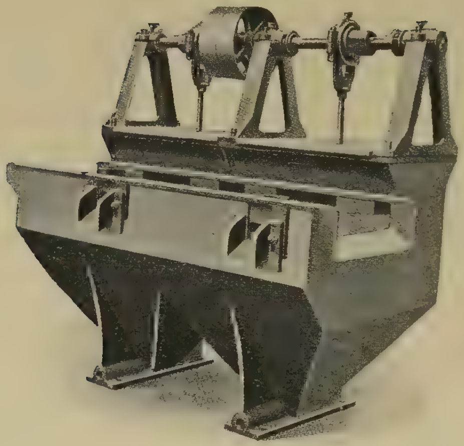

FIG. 224.

Two-cell Harz Jig.—General View. This illustration shows two gate discharges, one for each cell, and the final tailing discharge (p. 311).

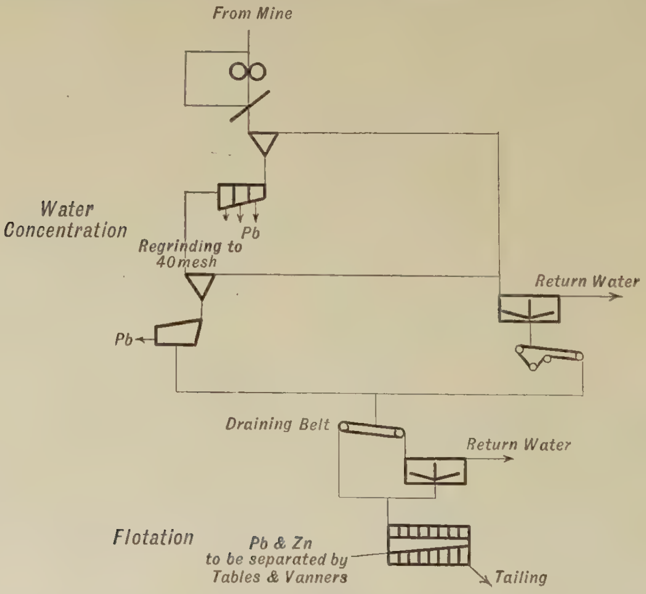

FIG. 225.

Water Concentration at the Central Mine, Broken Hill, 1913.—Flow-sheet. In this scheme, jigs, tables, and vanners, effect a water separation, to which is added a treatment by flotation, the floated concentrate in turn being water-concentrated to separate zinc from lead. Excluding the lead in the zinc concentrate and the zinc in the lead concentrate the recoveries were: lead 77·6 per cent, zinc 85·6 per cent, and silver 49·2 per cent. Of recent years this scheme has been altered, the jigs being retained but not the tables or vanners, ahead of the flotation plant (p. 314). (Fig. 315.) (Harvey, *Trans. I.M.M.*, Nov. 1918.)

however, has its own independent and adjustable water-service, while sometimes the amplitude of its stroke is capable of adjustment. In addition, the depth of the bed is under control by the operator, who can add to it or take from it as appearances indicate.

Working with such closely-sized material, the concentrates produced by these jigs are clean, but the multiplicity of sizing appliances and the great number of jigs, make the Harz system relatively expensive in screens, power, and labour. The capacity is generally about two tons per sq. ft. of total sieve area per day for a four-cell jig; it is greater for coarse material than for fine, and greater per unit of sieve area where the number of cells is less. The amount of water required is generally about five tons per cell for every ton treated, so that a four-cell jig would require about twenty tons of water per ton of ore. To minimize the consumption of water, progression and eventual discharge of the tailing is assisted by a gradient towards discharge; with coarse material the drop between two adjoining cells is generally 2—3 in.; with finer material a smaller drop suffices, and with sandy material one inch, more or less. The discharge of such sandy material is sometimes into a stay box through a slit in the end, this slit being an inch or so below the level of the water overflow; the water is thereby stayed, and unnecessary flow prevented (Fig. 215).

Practice in the Harz system is illustrated by the following representative statements:

At Clausthal the details of the separate jigs working on a lead-zinc ore were as follows:[1]—

Nature of Material.	Size of Material.	Size Aperture.	Strokes.		Method of Jigging.
			No. per Minute.	Length.	
	mm.	mm.			
Sized	22 –16	4	120	46	German
,,	16 –11	4	140	30	,,
,,	11 – 8	2	160	30	,,
,,	8 – 5·6	4–3	180	25	English
,,	5·6– 4		200	20	,,
,,	4 – 2·8		220	15	,,
,,	2·8– 2	3–2	240	13	,,
,,	2 – 1·4		260	8	,,
Classified	Sand No. 1	1	280	5	,,
,,	,, No. 2		300	5	,,
,,	,, No. 3		300	5	,,
,,	,, No. 4		300	5	,,

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[1] Heriot, *E. & M.J.*, September 11, 1915, p. 425.

At Halkyn, North Wales, treating a lead-zinc ore and using the English method entirely :

Jig No.	Nature of Material.	Size of Material.	Size of Aperture.	Strokes.					Bed taken from Jig No.
				No.	Length in Cells.				
					1	2	3	4	
		mm.	mm.		mm.	mm.	mm.	mm.	
1	Sized	8–5	8·5	182	28	14	25	19	..
2	,,	5–3	5·5	186	26	13	19	15	1
3	,,	3–2	3·5	186	25	14	23	10	2
4	Classified	2	2·5	230	20	15	17	11	3
5	,,	..	2·5	230	10	11	14	14	3
6	,,	..	2·0	210	6	6	10	12	3

At Broken Hill, the May jig, a type of Harz jig, separates a lead concentrate from a zinc middling in a three-cell primary jig, this middling, where the ore is fine-grained, being crushed for table and flotation concentration ; but, where the ore is coarse, for further treatment in a secondary five-cell jig making some lead concentrate, a middling for regrinding, and a quartz tailing. The heavy nature of the gangue prevents a zinc concentrate from being obtained by jigging (Fig. 225).

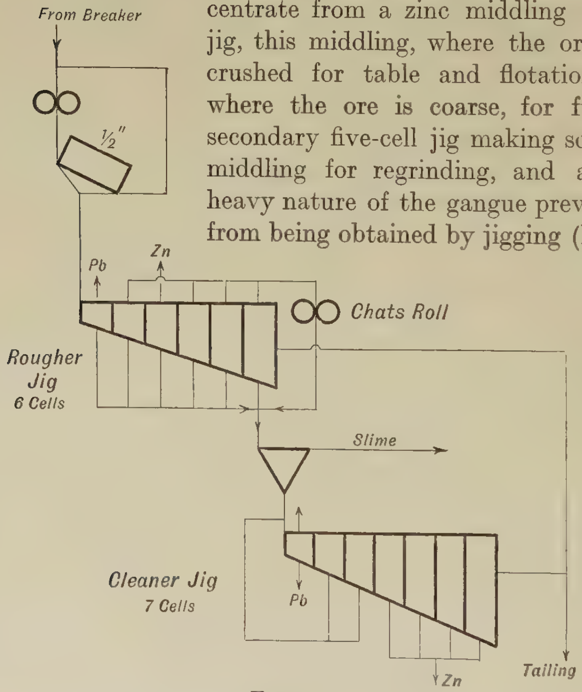


Fig. 226.

Joplin Two-jig System.—Flow-sheet. The ore carries about 5 per cent of zinc and a smaller percentage of lead in a chert and dolomite gangue (pp. 315, 379, 647).

The Joplin Jig.—With the Joplin jig the practice is entirely different from that just described. The coarsely-crushed ore is fed unsized to jigs which make a rough concentrate or middling, this in turn passing to other jigs which make a clean concentrate. Standard practice in the Joplin district is as follows : Crushed to pass a $\frac{1}{2}$-in. screen, the whole material, assaying about 4 per cent of zinc, goes first of all to a jig which discards final tailing while separating an enriched product assaying about 18 per cent for treatment on the next jig (Fig. 226). This first jig is hence described as a "rougher jig." It consists of five, six, or seven cells in series, the sieve aperture being about $\frac{1}{8}$ in. throughout. This large number of cells is necessary to ensure a clean tailing; the opportunity must be given again and again if the fine mineral is to be saved. The plunger stroke, generally of about $\frac{3}{4}$ in. amplitude, is made about a hundred times a minute. There being much fine mineral to separate, jigging through the sieve is practised, mineral and middling being drawn by a strong suction-stroke into the hutch, whence they are discharged as a single product, usually through open spigots. Clean mineral coarser than the sieve aperture remains above the sieve to form the necessary bed in the earlier cells; should it be more than sufficient for this purpose, the surplus is drawn off as a gate concentrate, or otherwise by remaining on the bed until worn small enough to pass there would be undue loss by attrition. In the later cells some middling may similarly be drawn off above the sieve, to be reground and then retreated.

The hutch-work separated by the rougher jig, in turn passes to the "cleaner jig," which is set to produce clean concentrate, assaying about 45 per cent, middling, and a further increment of clean tailing. With most of the coarse material already removed, the stroke in this cleaner jig is quicker and shorter, representative figures being 150 strokes per minute and $\frac{1}{2}$-in. amplitude. For the same reason the aperture of the sieve is smaller, being generally about $\frac{1}{12}$ in. On the other hand, with more careful work to perform, the cleaner jig has generally one more cell than the rougher jig.

Usually one cleaner jig will take the product from two rougher jigs, since these discard about four-fifths of the material fed to them. The capacity of such a rougher jig is about 8 tons per sq. ft. of screen area per day; that of the cleaner jig is substantially less, and about 3 tons. The capacity for a complete equipment on these lines is accordingly from 4 to 5 tons per sq. ft. of total screen-area per day.

Where lead is present in addition to zinc, a lead concentrate can generally be obtained from the first cell of both rougher and cleaner jigs, partly from above the sieve.

Joplin jigs are essentially large jigs, the ore-box being generally 3 ft. by 4 ft., though, sometimes, as when the Joust cell is employed, 4 ft. by 5 ft. With such dimensions the capacity of a rougher jig might very well be as much as 500 tons a day. The amount of water consumed per ton is much the same as with the Harz jig, perhaps a little less, but with so much ore passing, the work of the end cells is liable to be disturbed by the heavy flow, and devices for taking off much of this top water are sometimes employed. As the cleaner jig discards a proportion of tailing similar to that discarded by the rougher jig, the concentrate finally obtained weighs only about 4 per cent of the ore treated. Joplin jigging, it is seen, was developed on low-grade ore. The system is not so refined as with Harz jigs, but the installation being simple and the operating costs not great, its adoption is warranted where the ore will not bear more refined treatment.

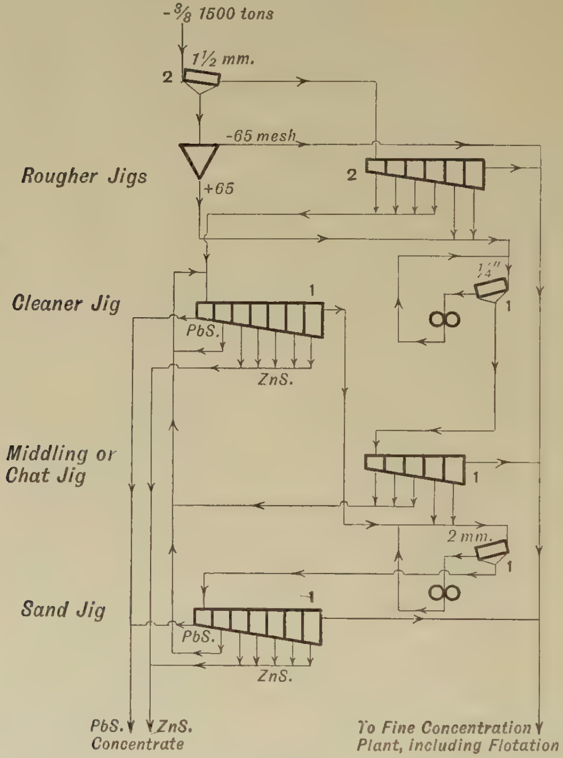


FIG. 227.

Joplin Jigs at the Netta Mill, Oklahoma.—Flow-sheet. The ore contains about 8 per cent of galena and blende together. In addition to the jigging plant shown, there is a table plant for the fine sand and a flotation plant for the slime. The capacity of this mill is 1500 tons per day, from which the recovery is about 70 per cent, or better. The block figures denote the numbers of the respective appliances for the above tonnage. It is seen that there are two rougher jigs to one cleaner (pp. 316, 647).

Where the ore is of better value and in sufficient quantity, the system is extended to include, it may be, 'middling jigs,' treating middling products from the cleaner jig, and 'sand jigs' to treat the fine material resulting from regrinding (Fig. 227). Nowadays, also, in addition to the recovery from jigs, the total recovery generally receives contributions from fine concentration and from slimes concentration, this latter being represented by flotation. On the other hand, where but the simplest equipment is justifiable or possible, a separation of clean concentrate may be made in a single jig.

Representative details of the different jigs employed in the Joplin system are as follows :

Jig.	Strokes.		Cells Number.	Sieve Aperture.
	Number.	Length.		
Rougher	100	$\frac{3}{4}$ in.	6	$\frac{1}{8}$ in.
Cleaner	140	$\frac{3}{8}$ in.	7	$\frac{1}{12}$ in.
Sand	170	$\frac{1}{4}$ in.	4	$\frac{1}{16}$ in.

The Joplin jig has not been widely employed, its use being more or less confined to the mines around Joplin, Missouri, and to those in the Wisconsin field. These districts are, however, so large and important, that the foregoing separate description of the practice is justified. Joplin jigging represents "roughing concentration."

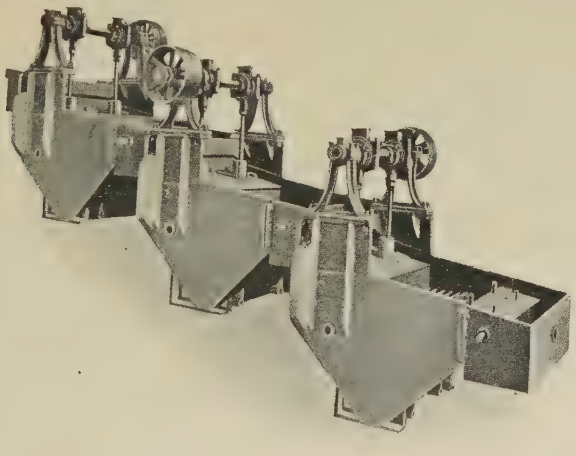


FIG. 228.

Three-cell Woodbury Jig.—General View. In this jig the plunger and sieve compartments of the cell are along the jig, and not across it as in standard designs. The plunger being relatively long, is supported and worked by two eccentrics. The shield which keeps the pulp from entering the plunger compartment is clearly seen (pp. 306, 318).

Jigging without previous sizing or classification is also practised with

Woodbury jigs at Lake Superior. There the ore jigged is that which has been crushed to pass a $\frac{1}{6}$-in. screen; it contains native copper. Three or four jig-cells go to make a series (Fig. 228). The first cell, using a 10-mesh sieve, is a classifying cell which separates slime, clean concentrate, and a hutch concentrate. Passing to the second cell with a 12-mesh screen, again two concentrates are made, the material passing on to the next cell, where, with a similar sieve, a middling is made, and, maybe, a tailing discarded.

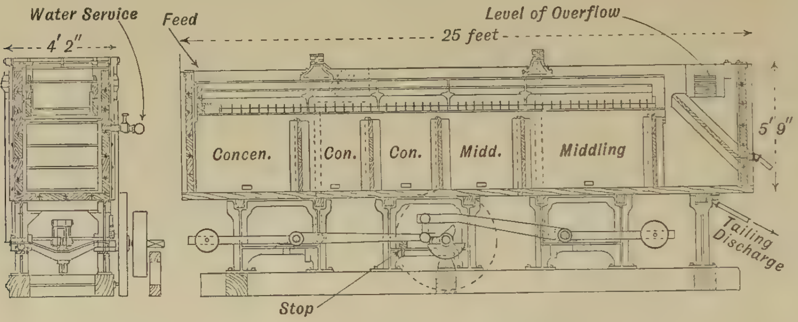


Fig. 229.

Hancock Jig.—Cross and Longitudinal Sections (Allis Chalmers). The tray is seen covering five compartments, in the first two or three of which, concentrate collects, to be discharged through holes or spigots at the bottom, while middling collects in the remaining compartments. From the end of the tray the tailing drops into the end compartment, the sandy portion being discharged below, with little water, the slime being discharged over a baffle, and the water overflowing through a hole in the side. In front of this last-mentioned hole is a gate, the height of which, and consequently the level of the water, can be regulated by slats. The ore arrives more or less dry, jigging being effected in water entered below the tray in the first three compartments.

On the tray are the cross slats which maintain the depth of bed. This tray hangs from two cross-bars, which, at their ends and outside the box, are supported upon upright rods rising from levers beneath. As by a cam these levers are lifted, the cross-bars are lifted also and the tray with them; but the first of these two cross-bars being held at either end by a link to a centre, cannot rise in a straight line upwards unless that link be horizontal. In any other position of the link the cross-bar must swing forward as well as upward, the second cross-bar and the whole tray following. With the passage of the cam the levers and the tray fall back by gravity, the main lever being brought up smartly against a stop (p. 318).

The Moving-sieve Jigs.—The only representative of this type of any present importance is the Hancock jig—formerly others were in use, the Bilharz, for instance. In the Hancock jig the ore-box consists of a long tray, 20 ft. in length by 2 ft. 8 in. in width, closed at the feed end and open at the discharge (Fig. 229). For bottom it has a series of sieves, of small aperture at the feed end, and of larger aperture at the discharge. This tray is suspended horizontally from upright side-rods (Fig. 230) in a large box or hutch, 25 ft. long, 4 ft. 2 in. wide, and 5 ft. 9 in. deep. By partitions rising from the bottom, this hutch is divided into five compartments under the tray, and a sixth or tailing compartment. The sieves in the tray have apertures to suit the respective products which it is desired shall be collected in the compartments beneath. Beginning from the feed end, the aperture covering the first compartment is generally about 3 mm.; that over the second compartment, 4 mm.; that over the third, partly 4 mm. and partly 9 mm., an aperture equal to that of the larger of the two sizing screens between which the feed was prepared; and that over the fourth and fifth compartments about 6 mm., finishing again with a narrow strip of 9 mm. Near the bottom of all these compartments are hutch discharges on each side, of which, as a rule, only those on one side are open at any one time, those on the other side being plugged. In the first three compartments water is added to the hutch under the tray, the ore arriving more or less dry. This water rising through the material collected on the tray overflows at the discharge end at such a level that the material is covered to a depth of about 3 or 4 in. In this situation the necessary jigging action is obtained from a cam below the box, which, through horizontal levers, raises the side-rods, these in turn raising the

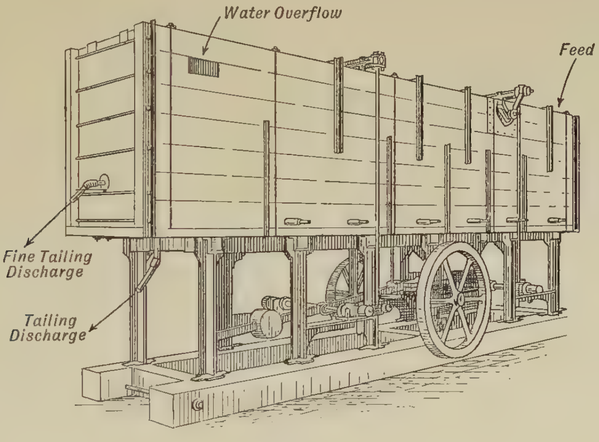


Fig. 230.

The Hancock Jig.—General View (Allis Chalmers). The cross-bars from which the tray hangs, the link which causes the forward and backward movements, and the rods rising from the lever-arms, are clearly indicated. The discharges from the compartments are also seen, the fifth compartment apparently having two discharges. There will be similar discharges on the other side of the box (p. 318).

tray, to let it fall again by gravity, the levers and with them the rods and tray being brought up smartly against a stop. To this up-and-down movement a to-and-fro component is added, by attaching the upper ends of the rods to links capable only of radial movement around a centre. The net result is that the tray makes a movement forward as it rises, and comes backward as it falls. By shifting the centre round which the link moves—this being possible along a slot—the two components of the complete movement, horizontal and vertical, may be varied, each from ½ in. to 2 in.; when one is given its maximum the other will be roughly at its minimum. The horizontal movement gives progression along the tray, while the vertical movement produces stratification; generally they are both about ¾—1 in. The number of strokes is 180—195 per minute; too slow a stroke will not be effective in stratification; too quick a stroke will carry the material forward at too rapid a rate.

The material fed to this machine is generally that which has passed a 9 mm. screen and been retained on a 2 mm. screen. Arrived on the tray, stratification is quickly accomplished by the strong pulsion-stroke as the tray falls back, the suction stroke being weak by reason of the depth of material on this portion of the tray. The material which descends into the first three compartments is generally clean concentrate. With the bulk of the clean mineral thus separated, it remains to recover middling in the fourth and fifth compartments, and to discard a clean tailing. In order that descent of the middling may take place the bed may not contain large heavy mineral. To ensure this freedom from heavy mineral, and at the same time to prevent such mineral from passing on eventually to find itself with the middling, there is over the end of the third compartment a narrow strip with large aperture. With the bed thus relieved, the middling can descend. A second strip of large aperture at the end of the tray similarly ensures that no large particle of middling passes into the tailing.

It is seen that the mineral in the ore itself forms the jig bed. To prevent this bed from creeping along the sieve, this latter is covered with a grating of coarse slats about 3 in. deep. Seeing that there is no provision for drawing off mineral above the sieve, nor any means of maintaining the bed regular in depth, this jig is only suited to the working of ores of low and regular grade. If more mineral arrives than is normal, the fourth compartment will probably deliver a concentrate, and its discharge on the concentrate side would be opened. If, on the other hand, less mineral were arriving, it might be necessary to shut the concentrate discharge of the third compartment, and open the discharge to the middling launder on the other side. Further, seeing that all the five compartments are fully occupied in making simple concentrate and simple middling, this Hancock jig is not capable of making more than a one-mineral separation.

It was designed and developed at the Moonta mine, South Australia, to treat a low-grade copper ore. From that beginning its use has

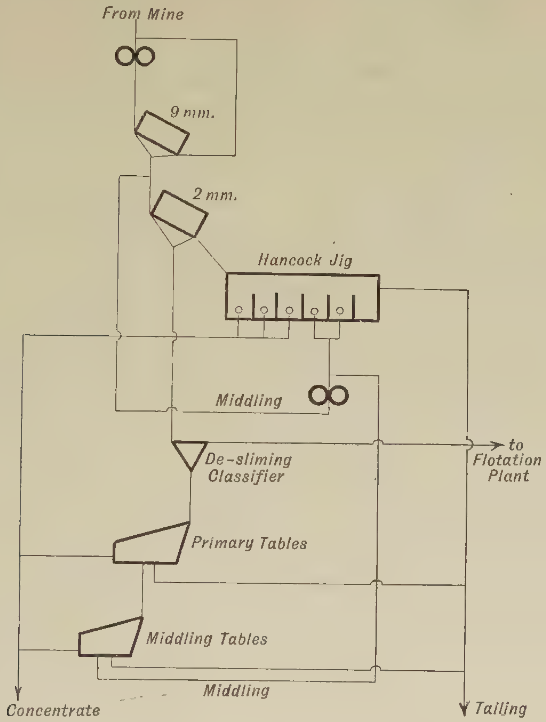


FIG. 231.

Coarse and Fine Concentration on the Missouri Lead-belt.—Typical Flow-sheet in 1916. Hancock jig followed by primary and secondary tables. Experiments have since shown that better results are obtained when the material passing the 9 mm. screen but remaining on the 2 mm. screen—that is, material which in the above scheme goes to the jigs –is crushed by ball-mill and then returned to the 2 mm. screen, with which the ball-mill is in closed circuit; with the jigs thus removed from the scheme, and the material ground finer, the increase in the amount of concentrate recovered from the tables is greater than the increase from the flotation plant (pp. 322, 376, 479, 481). (Delano and Rabling, *Trans. A.I.M.E.*, Aug. 1920.)

extended considerably, and to-day, though threatened by finer grinding, fine concentration, and flotation, this jig is greatly employed on disseminated lead mines in Missouri, on copper mines in British Columbia, and elsewhere. Tried at Broken Hill it was not adopted, the May jig giving better results.

Its great recommendation is that it treats material of a much wider range of size than the Harz jig and thus renders unnecessary a great number of sizing appliances. It has, moreover, a very large capacity, a standard machine being capable of treating 500 tons a day, or about 10 tons per day per sq. ft. of sieve area. In doing this it consumes only about 5 h.p., which is only about one-sixth of that which would be required by an equivalent Harz-jig equipment. It occupies much less floor space, and consumes very much less water, the consumption of this commodity being normally only about 7 tons per ton of ore, and less when the tailing is discharged by drag belt or shovel wheel. As indicated above, it is not suitable for rich material, partly by reason of the lack of provision for removing concentrate above the sieve, but also because rich ore fluctuates in value and the necessary attention to the bed which such fluctuation demands, cannot be given readily ; the inside position of the tray and its constant movement make such attention awkward. For rich material the Harz jig is decidedly more suitable, the beds of the separate cells being more readily and conveniently adjusted to changing circumstance. With low-grade material, however, the concentrate given by the Hancock jig is comparable in quality with that which might be expected from the Harz jig. On the other hand, in the endeavour to obtain a reasonably clean tailing, a rather bulky middling-product is made, necessitating considerable regrinding and eventual fine concentration (Fig. 231).

The Richards Pulsator Jig.—In this jig the jigging action results from impulses of water issuing under pressure from a water service ; there is accordingly no suction stroke. These impulses are obtained by placing a rotating valve on the water service by the jig hutch. Such water impulses being difficult of uniform distribution over an area, the ore-box is very small and generally only about 6 in. square. The whole jig-cell is just a cast-iron ore-box with feed hopper on one side, tailing discharge-lip on the other, gate discharge in front, and pointed hutch below, this latter terminating in a spigot discharge (Fig. 232). Behind is the rotating valve with its driving pulley, and an air-vessel to avoid water-shock. The number of impulses given is generally about 200 per minute, the water being under a head of about 10 feet. Such a pressure makes a bed 4—5 in. deep, satisfactory. Rising through this bed, the water quickly stratifies the material, no suction stroke interfering. The material must be closely sized, but with such material the capacity of this jig is large, and expressed in terms of unit screen-area about 100 tons per square foot per day. As with plunger jigs, this pulsator jig may be assembled in series, two or three cells generally making the complete jig (Fig. 233).

In spite of obvious advantages of small space, low power- and low water-consumption, this jig has not obtained the favour which might have been expected. It has been used with iron ores when, contrary to expectation, a good deal of hutch-work resulted, apparently from fine material falling between consecutive impulses. It was tried at Joplin, but did not there appear capable of adjusting itself to variations in the feed, the small bed having no steadying effect.

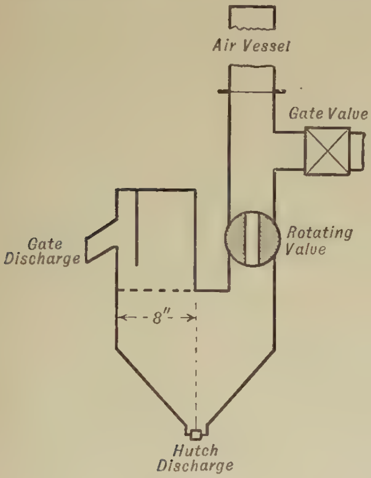


FIG. 232.

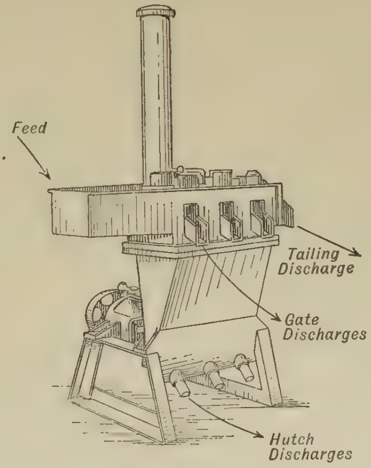


FIG. 233.

Richards Pulsator Jig Cell (Fig. 232).—Diagram. These cells are generally 6 to 12 inches square (p. 322).

Richards Pulsator Jig (Fig. 233).—General View of Three Cells in Series. Open gates for the concentrate discharge, and plugged spigots through which any hutch-work may be discharged, are seen (p. 322).

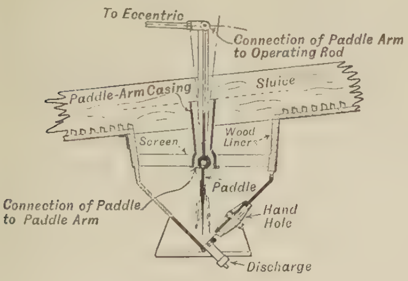

To Eccentric
Connection of Paddle Arm to Operating Rod
Sluice
Paddle-Arm Casing
Screen
Wood Liner
Paddle
Connection of Paddle to Paddle Arm
Hand Hole
Discharge

FIG. 234.

The Neill Jig.—Section down the Stream. This oscillating-paddle jig is for placing in sluice bottoms to separate and withdraw fine black-sands, etc., from auriferous gravels, or fine concentrate from stanniferous gravels (pp. 323, 348).

The Oscillating-paddle Jig.—The Neill jig of this type was designed for insertion into sluices to recover fine material escaping ordinary riffles and tables (Fig. 234). The water pulsation being obtained by a vertical paddle centrally disposed within the hutch, a plunger compartment is avoided, and practically the whole of the jig is useful sieve-area. This paddle oscillates upon a horizontal axis lying in the plane of the sieve and across the direction of flow. The actual oscillation is communicated by a rod rising from this axis upward through a sleeve projecting above the level of the material in the sluice, the upper end of this rod being reciprocated by an eccentric. This jig has been used on dredges recovering gold and tin, these heavy materials passing down through a bed of cast-iron shot placed upon an 8-mesh sieve.

Résumé of Jigging.—The main headings under which the principles of mechanical jigging have been described may be assembled into the following statement:

Methods of Jigging

German method; over the sieve.
English method; through the sieve.

Types of Jig Cell

Plunger-jig cell; plunger impulse and fixed sieve.
Tray-jig cell; tray impulse and moving sieve.
Pulsator-jig cell; water impulse and fixed sieve.
Paddle-jig cell; paddle impulse and fixed sieve.

Systems of Jigging

Harz jig; with close preparatory sizing.
Hancock jig; with wide preparatory sizing.
Joplin jig; without preparatory sizing.

The Harz jig is preferable when the close separation of two minerals has to be effected, or a rich ore treated. The Joplin jig, though capable of recovering two minerals, is not used where a close separation is required, or where the ore is rich. The Hancock jig is preferred only where one mineral is present and the ore low in grade but large in quantity.

Jigs connote coarse concentration; they take up the work of separation where hand-picking leaves it (Fig. 235). Like hand-picking they may make a clean concentrate, or discard a final tailing, or, as is more usual, they may accomplish both. Whereas, however, picking leaves the great bulk of the material to be sent to the mill for further comminution, with jigging the product needing further comminution should not be large, except the jig be used as a roughing appliance, that is, to make a bulky rough concentrate while discarding a worthless tailing.

The concentrate which jigs ordinarily give is attractively-clean mineral, cleaner generally than concentrate from machines treating finer material, though that good quality may be rather a function of the mineral-grain than an attribute of the jig itself; sometimes in obtaining this cleanliness much of the larger mineral is worn quite round, loss in fine abraded material thereby resulting. Ordinarily, also, the jig tailing is clean. Accordingly, it might very well happen that, for instance, a lead concentrate might assay as much as 75 per cent, and the tailing resulting from the same operation only 0·75 per cent.

In respect to the part they play in the total recovery, jigs where employed are generally responsible for the greater portion of that recovery, say, some three-quarters or more; but where their major purpose is to classify rather than separate completely, the amount of concentrate coming directly from them is substantially smaller. Altogether, the jig to-day remains a very important appliance. It is employed with all ores when the grain is coarse enough; notably, with silver ores at Cobalt, Ontario; with copper ores at Anaconda, Montana; with lead ores in Missouri; with zinc ores in Missouri and Wisconsin; with lead-zinc ores in the Rhineland and Silesia, in Great Britain and elsewhere; and with tin ores in Bolivia and South Africa.

Bearing in mind, however, the ultimate exhaustion of the richer deposits, and the coming ascendancy of poorer and finer-grained ores, it seems likely that the jig has been more important in the past than it will be in the future. Undoubtedly, at many individual mines it has been abandoned in

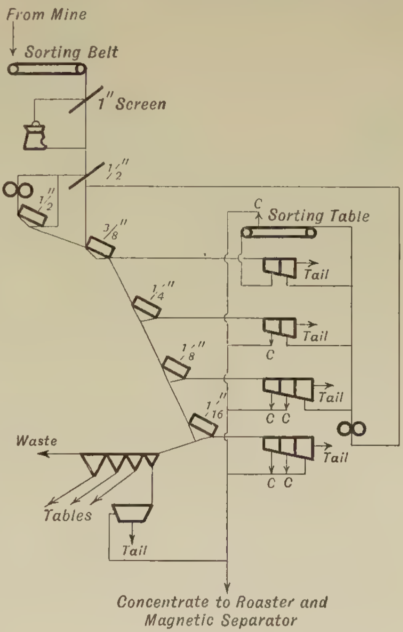


Fig. 235

Water Concentration at Storey's Creek, Tasmania.—Flow-sheet. The ore contains cassiterite and wolframite coarsely aggregated with some sulphides in a quartzose and granitic gangue. In this scheme the close association of jigging and sorting is clearly illustrated. It was possible to pick out pieces of clean wolframite before the ore was crushed, while after crushing it was still possible to pick out clean pieces of valuable mineral from the concentrate obtained from the first jig. The line drawn here between jigs and tables is that the jigs treat sized-material and the tables treat classified-material. All the jigs discard a final tailing. Each of the four spigot products from the classifier is tabled separately; the tabling of only the first spigot is shown (pp. 23, 324). (*Chem. Eng. and Mining Review*, Melbourne, Dec. 1917, abstracted *M. Mag.*, May 1918.)

favour of a fine concentrator, one or other of the shaking tables. In the earlier days of the porphyry-copper mines, for instance, jigs were included in the concentration scheme, whereas, to-day their place is taken by roughing tables. At Mt. Bischoff in Tasmania, where formerly 60 per cent of the tin output was obtained from jigs, to-day no jigs are working, tables having taken their place. Quite recently also, Hancock jigs, which not many years ago displaced plunger jigs on the Missouri lead-belt, have in turn given place to shaking tables. These changes in practice have largely been demanded by the diminished size of the mineral grain, but they have also been forced by the economic necessity of a better recovery only obtainable by finer grinding; the ore in all the above instances was simple. Where the ore is complex, as with lead-zinc ores, concentration must begin before the more intimate mixing which often follows fine-crushing has supervened, and jigging is retained. It is retained also where, as at Joplin, friable marcasite and pyrite would foul the entire concentrate if none were removed by jigs. Similarly, with tin ores in Bolivia and elsewhere, the very clean concentrate necessary to diminish the otherwise heavy freight to smelters, is more readily obtained from material as coarse as possible, that is, by jigging.

It is generally considered that jigs are not applicable when the mineral content is very low—say 1 per cent or less. Nevertheless, recently there has been some application of the Neill jig, to assist in the recovery of tin and gold from dredged and sluiced material containing still less mineral.

Other Vertical-current Separators

Willoughby Jig.—This jig so far has been used principally in Nigeria, to enrich low-grade alluvial tin-concentrate, bringing it to market value or thereabouts. In that country at times water is scarce.

The appliance consists of a water-tower about 5 ft. high and 20 in. in diameter, communicating by a horizontal passage with the bottom of an ore-box about 18-in. square and 12-in. deep. The whole, that is, water-tower, ore-box, and communicating passage, is fixed in a water-tank about 6 ft. long and of 30 in. square section, in such a manner that the water, issuing from the tower through a bottom valve and passing up through the ore-box, may overflow into the tank, and from the tank be hand-pumped back into the tower again. In that way the water is used over and over again with little loss (Fig. 236).

The operation is as follows: the crude concentrate, closely sized, is charged into the ore-box, the sieve of which is of suitably fine aperture, 20 mesh, 30 mesh, or finer, to support all the material. The water-tower being full, the bottom valve is opened, permitting the water to pass out and up through the ore-bed, this generally taking five or ten minutes. That valve being now closed, another valve below the ore-box is opened, whereby the water remaining in the ore-box may drain into the tank outside, the capacity of this tank being such that the water in the tower does not fill it to the level of the sieve.

During the upward rise of the water, the lighter mineral is brought to the top whence it may be scraped off, the operation being repeated until no more appears. The last stages of this removal are much facilitated by swirling the water above the bed, this swirling motion bringing the light material to the centre.

This machine, working intermittently, has only a low capacity, some 2 tons or so per ten-hour shift, but handling an enriched material, great capacity is not required.

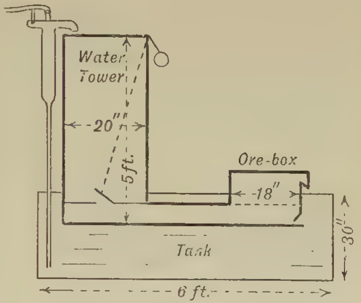


FIG. 236.

Willoughby Jig.—Diagram (p. 326).

Though particularly applicable to alluvial tin, which is granular, it may also be used on concentrate obtained from crushed ore, when fine hair sieves would probably be found more suitable.

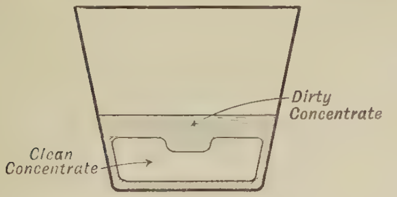


FIG. 237.

Kieve.—Diagram. The clear internal portion of the tossed and packed material is that which is relatively clean. The top layers can be removed by careful scraping; the central core into which much light material has been swirled, can be dug out; the outside skin around the sides and upon the bottom is left, because it contains impurities which did not come into suspension during tossing (p. 327).

Tossing Tub or Kieve.—The final operation to which tin concentrate obtained from the lode mines in Cornwall is submitted to make it marketable or the more marketable, is known as "kieving." In this operation a charge of about three-quarters of a ton of heavy concentrate is placed with water in a strong wooden tub or kieve, about 3 ft. in depth, 3 ft. in diameter across the top, and 6 in. less in diameter at the bottom, where, so far as possible, it is brought into suspension by vigorous stirring with a long-handled shovel or a mechanical paddle. When this 'tossing' is complete, a matter of a few minutes, the kieve is smartly rapped with a heavy bar or hammer while settlement or 'packing' takes place. By this knock, which is given about 100 times per minute, the settlement of the lighter particles is delayed, while the settled heavier particles exert their greater density to work themselves deeper into the bed; the result is that the top layer is of lower value. Removing it together with a central core, the desired clean concentrate may be dug out, care being taken to leave an inch or so of unclean material round the sides (Fig. 237). This operation, though simple, requires a good deal of experience to make it useful. Different concentrates require separate and different treatment, some indeed demanding that the tub should be held inclined. It is invariably used in Cornwall, where by its use the tin concentrate is raised from, say, 45 per cent of tin to 65 or 70 per cent; or a wolfram concentrate is similarly enriched. On the Welsh zinc and lead mines the kieve serves a similar purpose.

It is probable that kieving, which uses simple fall in water, is so successful in Cornwall because it is there largely applied to concentrates which, coming directly from appliances known as buddles, are to a large extent sized products.

FINE CONCENTRATION AND SLIME CONCENTRATION; HORIZONTAL-CURRENT MACHINES

As when describing comminuting machines, here also it would be inconvenient to reserve separate descriptions for sand machines and slime machines. Whether for one class or for the other, the material treated is essentially fine; in addition, being almost invariably classified-material, the mineral particle is much smaller than the gangue particle. That being so, and the necessity now being a sizing action only possible by water streaming over a surface, practically all the fine separating machines use stream- or film-sizing.

It has already been shown that the stream-sizing of classified material results in an upstream position for the mineral and a downstream position for the gangue (p. 257). Unless the previous classification—water-sizing in vertical currents—had been extraordinarily close, there would be, in addition to mineral and gangue particles of different size, particles of equal size. Considering these, it is obvious that by reason of the greater pressure of the mineral particle upon the bottom that particle would lie upstream, and the gangue particle downstream. Undoubtedly also there would be particles of the same density but of different size; of these, the small would remain upstream while the large would move downstream. Finally, there would be large particles of mineral and small particles of gangue. Considering these last, if the respective frictions with the bottom determined their positions, the relatively large particle of mineral would remain while the small particle of gangue would be swept away. The overriding circumstance undoubtedly is, however, that the larger particle by projecting into swifter-moving films is rolled away in spite of its greater pressure upon the surface. In all, then, the disposition resulting from this stream action would be: the mineral upstream in respect to the gangue; the fine particle upstream in respect to the coarse; and mineral and gangue overlapping along a zone where coarse mineral finds itself associated with fine gangue (Fig. 166). In any dressing machine, the mineral upstream would be the concentrate, the gangue downstream the tailing, while between the two would be a middling having a nature the reverse of classified, that is to say, a middling of counter-classified material.

The possibilities of separation by streaming being as described, it is not surprising that this movement of water is largely employed. With many machines, and particularly with the earliest and the simplest, it is the only movement. With others, forming a second class, an oscillating movement of the appliance is superimposed. While, finally, with others of a third class, a jerking or throwing movement is added to streaming. Under these classes the various appliances may be described.

FILM-SIZING AND STREAMING CONCENTRATORS; FRAMES, STRAKES, BUDDLES, SLUICES, ETC.

From what has been said concerning the important part the surface for deposition plays as a drag upon the water and material streaming over it, it is reasonable to group the various appliances in this and in the other two classes, according to the nature of the depositing surface they respectively present. Such surfaces may be divided into:

1. Plain or smooth surfaces, where retardation of the material is effected principally by friction with the surface, and
2. Riffled surfaces, where, in addition to friction, positive retardation is effected by riffles or stops.

Before proceeding to the descriptions of appliances in detail, some remarks upon the transportation of material by streams will perhaps be helpful. Transportation in a stream takes place partly by rolling, this being the behaviour of the coarser particles, partly by leaping, as with intermediate particles, and partly by suspension, this being the circumstance of the finer particles. Further, where the bottom is rough, the tendency to roll is greater, while where smooth, as on glass for instance, the tendency to slide is more pronounced. In a deeper and faster stream, of course, gravel and even boulders may be rolled along.

Some idea of the velocity necessary to transport material of a given size may be gathered from the following considerations: the drag on the particle tending to move it along is fluid friction, while the resistance is a simple function of the particle weight, the solid-frictional function. But fluid friction—in the shape of eddy resistance—and particle weight, are the two factors determining the fall of a particle in water, a subject exposed when discussing the principles of water-sizing (p. 250). In that exposition, it was seen that the resistance to fall arising through fluid friction was generally accepted as varying with the square of the velocity, and that, in the end, the velocity of fall was proportional to the square root of the product of particle diameter multiplied by the particle density diminished by the density of the medium. Further, that for quartz falling in water the velocities were given by the formula:

$$v = 112\sqrt{D}$$

where v = millimetres per second,
D = diameter in millimetres.

From that formula, with a diameter of 1 mm. the velocity of fall, or in other words the upward velocity necessary to support the particle, would be 112 mm. per second. If the diameter were 1 in., or 25·4 mm., the required velocity would be five times greater, that is, 560 mm. per second, or 1·83 ft.; if it were 4 in. the velocity would be twice that figure again, or 3·66 ft. per second; and if it were 9 in., it would be three times 1·83 ft., or 5·5 ft. per second. Velocities slightly in excess of these would move the respective particle upward.

To move the particle along an approximately horizontal plane would appear to require lower velocities since the force to be overcome would only be μW, a fraction of the weight, μ being the coefficient of solid friction. But the lowering of velocity will not be so great as might be imagined. If, as shown, the velocities vary as the square root of the particle diameter, they will, seeing that weight varies as the cube of the diameter, vary as the sixth root of the particle weight. At that rate, taking the coefficient of solid friction to be about 0·33, the sixth root of which is about 0·83, the velocity necessary to move a given particle horizontally would be 0·83 times that necessary to lift it vertically. In view of this, and seeing that the layers most active in transportation, the lower layers of the stream, move at a slower rate than the upper layers, it may be considered that the formula for the velocity of fall of particles in water gives very fairly the order of stream-velocity required to move given particles horizontally. To move them at an adequate rate requires greater velocity. At an average stream-velocity double that calculated to be necessary to move the largest piece, it may be considered that a mass of mixed sizes travels at about half that average velocity.

That the capacity of a stream to transport material varies as the sixth power of its velocity may also be deduced directly from stream conditions, thus: μW the force of friction to be overcome is equal to $\mu D^3(\delta - 1)\omega$; the drag on the particle tending to move it along varies as the wetted surface and as the square of the velocity, being K^1Sv^2, or KD^2v^2. When drag and friction are equal, that is, when $\mu D^3(\delta - 1)\omega = KD^2v^2$, the particle will be about to be carried along. Leaving out the constants, D^3 will therefore vary as D^2v^2, and D as v^2. But W varies as D^3, and, accordingly, W, the weight transported, will vary as v^6. If fluid friction were taken to vary as $v^{1\cdot 83}$ in place of v^2, as recent determinations indicate, W would vary as $v^{5\cdot 49}$.

A stream has greatest capacity to transport when no water flows unconcernedly away above the material borne along the bottom. Accordingly, the proper depth of a transporting stream is largely determined by the size of material to be transported; any depth of water above the moving material is out of action. On the other hand, the depth may not be so shallow that the larger pieces project above the water.

Transportation and treatment for the recovery of mineral are, however, very different things, the latter demanding the retention of the mineral. The transport of slime, for instance, sanctions a relatively deep stream, whereas the treatment of slime demands but the thinnest film. Treating large material, as in sluicing, the water-depth and velocity must need be great, for transportation; the bottom must then provide protection for the fine mineral-particle which, in the rough stratification set up, occasionally finds itself in contact with the bottom; great length of sluice is also required that the number of these occasions may be multiplied. Treating slime, on the other hand, the force of the stream is so adjusted that the mineral remains on the gently-inclined plane while the gangue is removed; moreover, the film of water is so thin that the mineral-particles are practically never off the bottom; no great length of appliance is therefore necessary.

Inclined Table or Rectangular Frame.—Beginning with the simplest machine and that treating the finest material, the inclined table or flat frame is used, largely in Cornwall but also elsewhere, to treat the slime overflowing from classifiers, that is, material the great bulk of which will pass 200 mesh and the whole of which will generally pass 120 mesh. This appliance consists of a flat table about 6 ft. in width and in length, made of soft-wood boards, yellow pine for instance, placed crosswise, and laid, at an inclination of about $1\frac{1}{2}$ in. per foot, below a distributing board over which the pulp arrives (Fig. 238). This pulp, having been thickened to a consistency of about ten of water to one of ore, streams over the table with a force sufficient to carry away the lighter particles while permitting the denser particles to settle; the velocity of the stream is about one foot per second, and its depth about one millimetre, with a double depth at the wave-crests. The greater velocity necessary for larger material is obtained either by greater inclination or by greater volume of water. Settlement proceeds till the whole surface is covered with a thin deposit, this usually taking about four minutes.

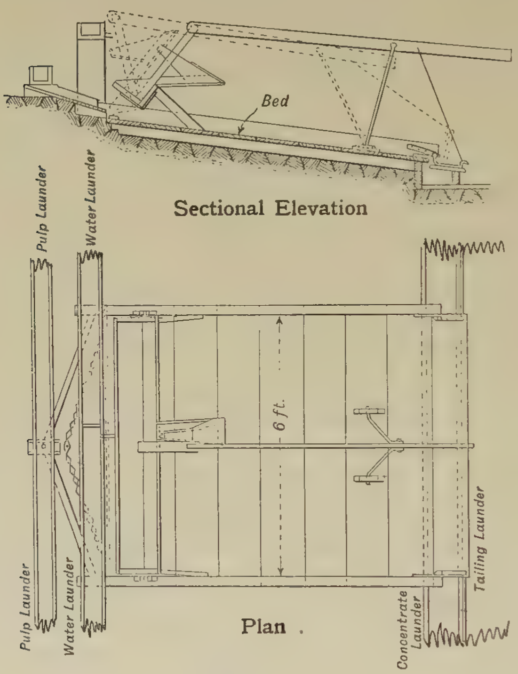


FIG. 238.

Inclined Table or Rectangular Frame.—Sectional Elevation and Plan (Holman Bros.). The bed is made of cross boards laid at an inclination of 1¼ to 1½ inch per foot. At its head comes the distributing board with distributing buttons, and the pulp launder; at its tail, the hinged tailboard or lap which conducts the tailing to an outside launder, but acts as a stop to the downward-flushed concentrate, directing this concentrate into an inside launder. The water balance, by which this changing position of the tailboard is effected, through rods, rockers, etc., is clearly shown (p. 331).

Meanwhile, the undeposited portion of the pulp, together with the material in suspension, flows off the table over a tail-board into a launder. Then, usually without any washing, the collected material is flushed off by additional water, which uninterruptedly has been filling into a balance box spanning the head of the frame, this water, overbalancing its box, pouring upon the table. Were the tail-board still in position there would be nothing to prevent the concentrate from following the tailing into the same launder; but the overbalancing of the water-box has, through a rod and chain, so raised this tail-board that discharge into the tailing launder is blocked, while passage is opened for the concentrate to shoot back into a second launder running parallel and alongside the tailing launder. This occupies but a moment, and, the water being away, the balance-box returns to its original position under the water service.

The amount of material which can be treated by a single table is about half a ton per day, or, say, one-tenth of a ton per foot of width.

The quality of the work done as shown by figures of recovery and enrichment, is not good; but it has to be remembered that, owing to the impalpable fineness of much of the material, some undoubtedly being suspensoidal if not colloidal, the work allotted to this appliance is extraordinarily difficult. In Cornwall the concentrate recovered generally assays about twice as much as the material fed, while the mineral contained in it is about half that originally in the feed. Such a concentrate must of necessity receive further treatment. Better results would be obtained with tin ore if the table were only about half the length stated, this length of 6 ft. being that demanded by the more easily-moved sulphides of lead and zinc. The principal purpose of this appliance, however, is not to effect great enrichment, but to discard a worthless tailing.

With such a low capacity many are required. In Cornwall, accordingly, these frames, known as "ragging frames," are assembled in long rows, back to back, sometimes in hundreds, this assembly constituting what is described as the 'Slimes Plant.' Being cheap and occupying much floor space these ragging frames are left in the open, generally below the settling pits which serve them.

Revolving Tables or Round Frames.—Greater capacity and a washed and therefore cleaner concentrate, are obtained by a circular table applying the same streaming action. Of such round tables there are two types, namely, the concave table, of which the bed slopes inward from the periphery, and the convex table, where the bed slopes outward from the centre. The latter may be considered to be the more general design, though in Cornwall the concave frame is more often seen. Both are supported upon an umbrella framework radiating from a central spindle by which the appliance is driven—generally from a worm and worm-wheel above but

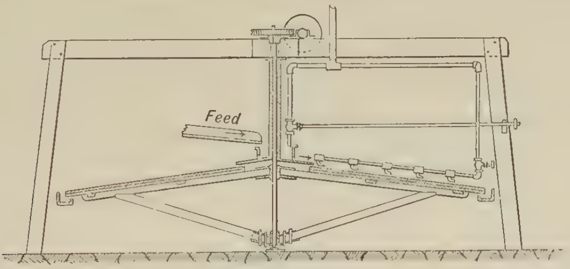


FIG. 239.

Convex Revolving Table.—Sectional Elevation. The illustration shows plainly the umbrella framework and driving gear, the central feed and wash-water service, and the special jets for the removal of the collected concentrate (p. 333). (Commans, *Proc. Inst. Civ. Eng.*, Vol. CXVI., 1893–94.)

often by bevel gearing (Fig. 239). At the centre of the convex table is a stationary distributing head with conical surface; or there may be a curved distributing launder. Following this head and after a small drop, comes the revolving bed itself, an annular surface. Feeding of the pulp takes place through an arc which may be as little as 90° or as much as 270°; continuing round the circle comes the stationary water-service, delivering wash-water through most of the remaining circle. Round the periphery is the stationary receiving-launder into which all the products duly fall, each into its properly-partitioned section, the tailing first, the wash-water tailing or middling farther round, and the concentrate last, complete removal of this concentrate being effected by special water-jets, a reciprocating brush, or a stiff wiper (Fig. 240).

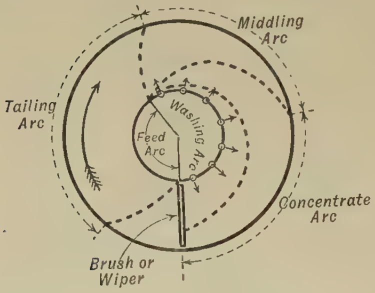


FIG. 240.

Convex Revolving Table. Diagram illustrating Manner of Separation. In the illustration the feed is shown continuing through an arc of about 140°. Throughout that arc the gangue particles stream radially down the bed, but since that bed revolves the path taken by the particle in respect to the stationary launder, is one curved forward in the direction of revolution; the tailing launder must accordingly be moved forward to pick up the end of the curved path taken by the last-fed particle of gangue. The middling product, moving more slowly down the plane, gets farther round the circle before it reaches its launder; the curve of its path is therefore more pronounced. While, finally, the concentrate, clinging more tenaciously, would in part go round the circle again were it not removed by a brush or wiper, or by special water jets (p. 334).

The concave table is very similar; with it, however, the pulp is fed round the periphery, whereas the products are taken off by an annular launder at the centre, this launder being divided by the necessary stops to keep those products separate. From their reversed design, the two types do not operate with quite the same effect. With the convex table, the stream spreading from the centre to cover the wider area at the periphery, is relatively strong at first and weak afterwards; in this design, therefore, the material which collects at the centre will be relatively clean and of small bulk. With the concave table, on the other hand, the stream velocity being least at the periphery where the space for settlement is greatest, a poor but relatively bulky concentrate results. These two appliances may accordingly be used, the concave as a rougher machine, collecting material to be treated by the convex, as cleaner.

The standard round table in Cornwall is the concave table of 18 ft. external diameter, and 8 ft. internal diameter (Fig. 241). Such a table, receiving pulp consisting of about eight of water to one of ore, through an arc of 270°, will treat five tons per day—or about one-fifth of a ton per foot of feed arc—making a concentrate about four times richer than the feed, and containing about two-thirds of the mineral originally present in that feed. It does this work while revolving at a speed of about one revolution in four minutes; the duration of the depositing period, in other words the time taken in passing through the feed arc, is accordingly three minutes. The deposited concentrate is then washed as it passes through an angle of about 70°. The quality of the material collected in the launder below this washing angle depends largely upon the slope and length of bed, and the amount of wash-water. If the slope has been proper, say $1\frac{3}{4}$ in. per foot, the bulk of the poor tailing will have been removed before washing begins, and, if desired, the product removed by washing may be regarded as middling.

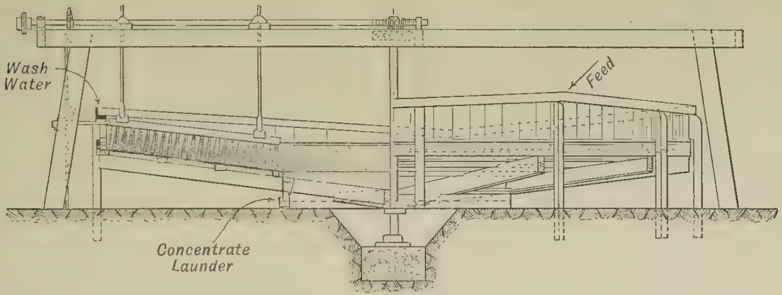


Fig. 241.

Cornish Concave Revolving Table.—Part-Sectional Elevation (Holman Bros.). The feed arrives at the highest point of a peripheral distribution, from which it flows both ways round the circle to embrace an arc of about 270°. About 70° of the remaining circle is devoted to wash-water, which is separately fed near the end of the circle, flowing thence in a backward direction around the circle till wash-water and feed meet, this occurring at the lowest point round the periphery. The uniform division of the feed and wash-water is effected by distributing buttons similar to those shown on the rectangular frame (p. 334).

The round table is a decided advance on the rectangular frame both in capacity and in the quality of the work done. Being a more costly appliance it is placed under cover and supplied with a feed of somewhat thicker consistency, this feed being best prepared in settlers with continuous bottom discharge, conical or pyramidal settlers, for instance. The opportunity to wash is one of its advantages. By washing, there is opportunity to remove the gangue arriving last, that is, fed nearest the line of concentrate removal, while if desired, by increasing the wash-water, such a middling product may be washed out as shall leave the concentrate much enriched. Washing also is a means of correcting irregular conditions of the feed, more wash-water being given if the feed is thick, and *vice versa.* These services it renders at little loss of mineral, but only at the expense of the water used. Ordinarily, in Cornwall, less than half as much water as that which enters with the pulp is used in washing. In the removal of the concentrate another and larger amount of water is usually necessary.

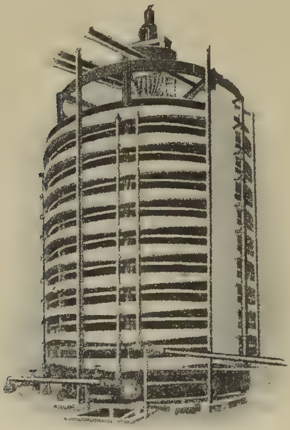


FIG. 242.

Multi-deck Revolving Table.—General View. In the concentrator illustrated, each deck consists of a concrete surface upon a steel framework. The decks are 18 feet in diameter, and one foot apart vertically; they are carried upon steel columns at the periphery, and these upon a steel ring, which in turn is supported on wheels running on a circular track. At the centre of the structure is a ladder-way permitting access to all decks and feed launders. The rate of revolution varies from one revolution in 3 minutes to one in 10 minutes. The pulp generally consists of 8 to 12 parts of water to one of solid, 5 to 10 tons of the latter being treated per deck per day (p. 336). (*M. & S.P.*, Aug. 8, 1914.)

Similar revolving-tables have been widely used elsewhere; upon tin ores, as at Mt. Bischoff, Tasmania, and in Bolivia; upon native copper ores at Lake Superior; and upon lead-zinc ores on the Continent and elsewhere. Before the advent of flotation they were the best appliance for recovering the finest mineral. As recently as 1916 such round tables were still used to recover copper sulphides at Anaconda, Montana, where to minimize the extensive floor space otherwise necessary, ten or a dozen tables, each a separate and complete appliance, were decked one over the other (Fig. 242). Similar 'multi-deck' tables with a smaller number of decks have been used elsewhere. Rectangular tables have also sometimes been assembled into multi-deck machines, but not with any great measure of success.

A difficulty with these round tables is the maintenance of the true running necessary for their proper functioning. If these tables do not keep their plane throughout the revolution, unsatisfactory working obtains. The ordinary wooden construction is not stiff enough beyond a diameter of about 18 ft. The multi-deck tables mentioned above were not supported at the centre but upon rollers at the periphery. In another design, the Linkenbach table—used considerably on the Continent for the recovery of fine lead and zinc sulphides—the bed is stationary while the distributing and collecting services revolve (Fig. 243). Obviously the same result is obtained, since the essence of both designs is the relative movement between the inclined bed and the respective launders. There must, however, be other stationary launders, one for each product, into which the collecting launders deliver.

Revolving tables being balanced machines require but little power, one horse-power being generally a sufficient provision for five or more.

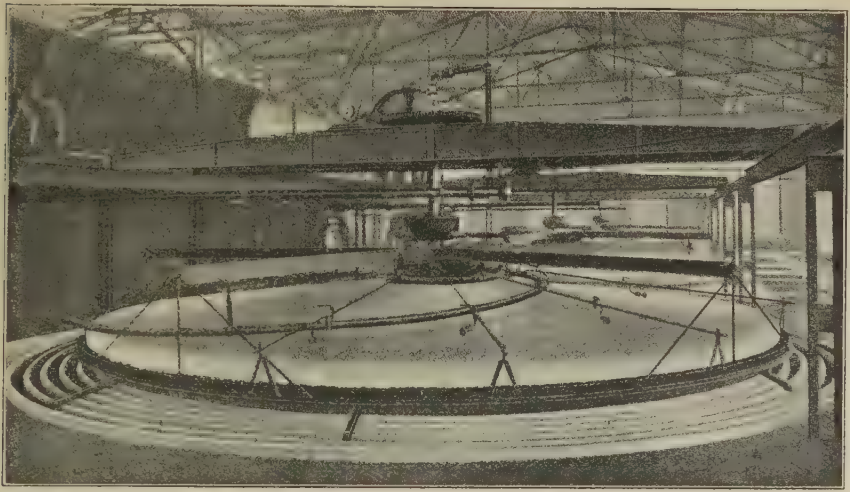


Fig. 243.

Round Table with Stationary Bed: the Linkenbach Table.—General View. The thickened slime from the pointed boxes faintly seen in the left upper corner of the picture is fed into a revolving segment at the centre, being therefrom spread over every portion of the bed in succession. Following this around the circle comes a wash-water service which, by means of jets, gradually cleans the deposit left after passage of the feed; and then, in turn, a more copious water-supply to flush off the cleaned concentrate. These water-services are arranged by pipes and jets over the bed, the illustration showing very clearly the spiral pipe from which the flushing water issues.

Revolving round the periphery comes, first and quickly following the feed, the tailing section of an annular revolving-launder, this section discharging by means of a projecting pipe into the inner of a series of fixed annular launders; then a middling section leading into a middle launder; and finally the concentrate section leading into the outside launder (p. 336).

The wooden surfaces of the tables so far described are sometimes replaced by cement, sand-concrete, rubber, a coating of paint, sand, etc., but never with any advantage to recovery so long as the wood remains in good condition. Cement surfaces cost more at first, but less afterwards; they are a feature of the Linkenbach table.

Blanket Strakes.—The gently inclined surface may, however, sometimes be advantageously covered with blanket, this constituting a blanket strake. Such a strake is a long shallow trough of width a little less than that of an ordinary single blanket, so that the blanket overlaps the sides. The length of this trough will vary with the purpose to be served ; if the blanket strakes are to recover fine mineral from an impoverished pulp as this streams away, they will be long ; but if they are set to recover, for instance, coarse unamalgamable gold from a battery pulp, the short length of a single blanket suffices. When the strake is long, the lower blanket is laid first, and then, overlapping it by about 6 in., the second, and so on. Two strakes are generally placed alongside, one being in operation and the other in process of being cleaned. In operation, the pulp flows over the blanket until the pores and fibres are filled and the surface loses its roughness; no manual help is given during this period of deposition, the pulp takes its own somewhat-irregular course down the bed. Then, the stream being turned into the other strake, the blankets are carefully folded and removed to a tub, where they are washed ; by the time they are relaid—generally some two hours—the others have become loaded. The concentrate recovered, though often only a small portion of the mineral in the pulp passing, is of relatively good grade. Such strakes have been widely employed to recover fine auriferous pyrite from departing tailing ; to recover friable copper sulphides escaping with the stream ; and exceptionally, to recover coarse gold when such has escaped the amalgamating plates, or being unamalgamable has not been passed over such plates. At the Ouro Preto mine, Brazil, for instance, a substantial proportion of the output is obtained from blankets, amalgamation not being practised.

Canvas Tables.—A similar service to that just described is sometimes rendered by tables covered with canvas instead of blanket. Canvas tables are as a rule wider than blanket strakes, but not so long. The canvas being fixed and not removable like the blankets, the operation is a little different. First, comes the period of deposition, which may last twenty minutes. Then, the pulp being turned off, the deposited material is lightly brushed with a broom as clear water streams over it ; thus is washing accomplished. Finally, comes the removal, which is effected with water from a hose, the concentrate being swept to the bottom.

Canvas tables have been used extensively upon material too poor, by reason of previous treatments, to pay for any more elaborate treatment. They are a feature on the Mother Lode, California, where they follow vanners. They are also effective when such friable ores as tungsten ores are being treated. Exceptionally, at St. John del Rey, Brazil, the greater portion of the gold recovered is obtained from canvas tables, ordinary amalgamation being impracticable because of abundant arsenopyrite.

Canvas Belt Concentrator.—To make a continuous operation of that performed intermittently on an ordinary rectangular table, Brunton arranged an endless cloth to move around two end rollers, climbing up a slight incline to reach the higher roller, then turning over to dip into water before returning to the lower roller (Fig. 244). On to this cloth the pulp was fed about a third of the way down the incline, the cloth being supported upon a wooden platform, over which it moved. The upward movement of the cloth carried the mineral with it, while the gangue was washed downstream. Passing upward beyond the point of feed, the settled mineral was cleaned by water jets, the remaining concentrate passing on to be detached as the belt rolled

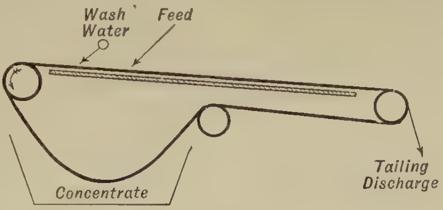


FIG. 244.

Canvas Belt Concentrator.—Diagram of Working (p. 339).

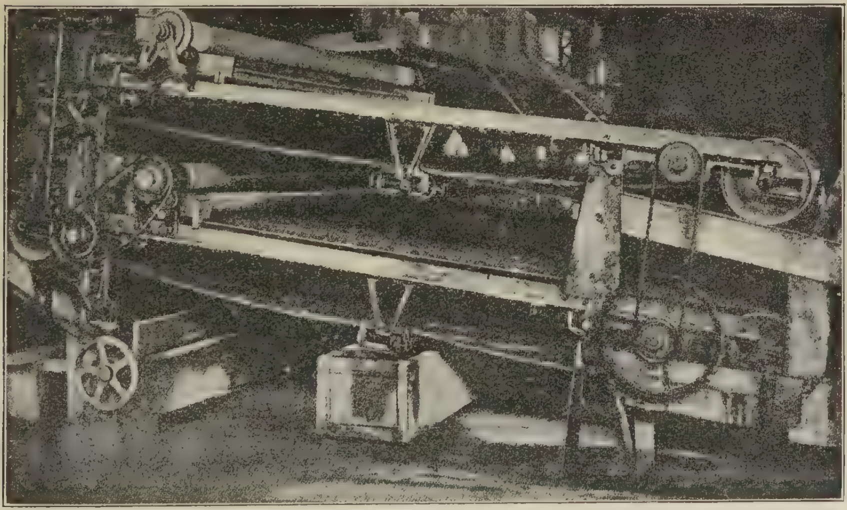
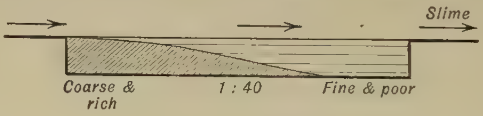


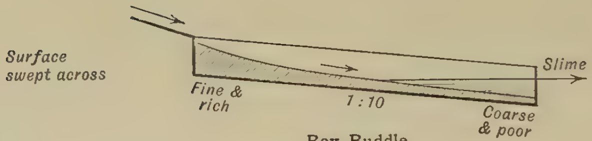
FIG. 245.

Canvas Belt Concentrator.—General View. Double concentrator used at the Steptoe Valley Concentrator, Nevada. Width, 8 feet; length of incline, 13 feet; material treated all below 200 mesh; consistency, 5 : 1; assay of feed, 1 per cent copper; assay of concentrate, 6 per cent copper; recovery of copper, 30 per cent; two belts used in series treating 8 tons per day (p. 339). (*E. & M.J.*, Aug. 17, 1912.)

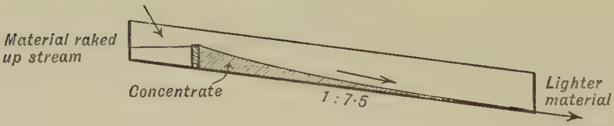
over into a water-bath. These cloth belts were originally used upon some lead mines in the North, but similar belts have also been used to save copper sulphide in the United States (Fig. 245). In other designs the belt has been side-inclined, discharging its products over the lower edge, the concentrate, however, sometimes over the end.



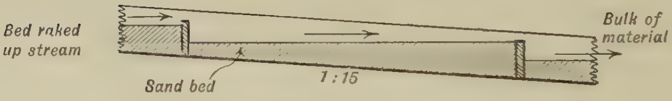
Strip or Box Settler.



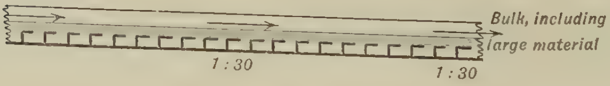
Box Buddle.



Tye or Cleaning Box.



Sluice Box, with stops.



Sluice Box, with riffles.

FIG. 246.

Box Concentrators.—Diagram of Strip, Buddle, Tye, and Sluice (pp. 340-348).

Tye or Cleaning-box.—The appliances so far described have been those with a relatively plane surface. Between these and those with a riffled surface there are one or two appliances using the rough surface of a collected bed of mineral. Of these, the tye or cleaning-box is a wooden box about 12—16 ft. long, 2—3 ft. wide, and 12—18 in. deep, in which a granular concentrate, generally obtained from sluices presently to be described, is dressed to a marketable condition (Fig. 246). Into the upper end of this box—which itself is laid at an inclination of about 2 in. per foot—the material is charged, to be carried by a stream of added water over a stop into the main portion of the box. Down the regular surface so formed the lighter particles are carried, while the valuable material is, by hand, continually raked up against the stream, till, at the head, a clean concentrate has accumulated.

In the Malay States rich stanniferous alluvial material is treated in similar boxes. These are coffin-shaped, one of small size being 1 foot wide at the head, 2½ ft. wide three feet down, and 15 in. at the lower end, the whole box being nine feet long. Such a box has a depth of about 1 foot; it is closed at the head and open at the tail. At the point of maximum width there is a stop about 4 in. high. Treatment takes place below this stop, the concentrate collecting there till the top of the stop is reached, when the box is cleaned up. In such a box about 100 tons of stanniferous gravel can be washed in a shift. Larger boxes of 25 ft. total length and 4 ft. maximum width are used where the scale of working is larger.

Buddle.—The rectangular buddle, formerly used in Cornwall, is a similar appliance (Fig. 246). Over a distributing board a heavily-laden mineral pulp flows into a box about 18 in. deep, 4 ft. wide, and 6 ft. long, where it spreads over mineral already collected there, the surface of which is kept even by a broom. Here, however, there is no raking of the mineral against the stream. The heavy material settles at the head, and the light and sometimes worthless material at the tail, leaving between them a middling worthy of further treatment. At the bottom, through holes in the tail-board to be plugged as the level of the collected material rises, the water and slime flow away. This box buddle is thus a collector as well as a concentrator; when full, the separate products are dug out.

In a simple way, and at the cost of a good deal of handling, the box buddle did good work, which, repeated, eventually gave a product such as, after final treatment in the kieve, could be marketed. Its capacity, however, was very limited, even for those early days, and in due course the round buddle became the adopted practice. Round buddles, like round frames, may be either convex or concave. The convex buddle has at its centre a conical distributing surface known as the centre head (Fig. 247). From this centre head, which is about 5 ft. in diameter, the mineral-laden pulp drops about a foot or fifteen inches on to a flat cement bed about 5 ft. long, making the total diameter 15 ft. At the end of this bed, and all round the buddle, is a tail-board about 12 in. in height; or more commonly the buddle is sunk in the ground to that depth. At one place in this periphery the tail-board is pierced with holes capable of being plugged as the collected bed rises, the open rows above permitting the slime to escape in suspension. In the annular bed thus provided, the concentrate collects at the head, its finer portion close up against the centre, while the rougher and poorer material collects at the periphery. While this deposition is taking place, the surface is kept regular and channelling is prevented, by sweeps of brushwood, of yarn, or of cloth, dragged slowly round by revolving arms. The collecting material makes its own inclination and its own bed, a bed which is steep with coarse material and almost horizontal when the material is very fine. With coarse material, brushwood sweeps would be used; with fine slime, sweeps of the lightest of rags.

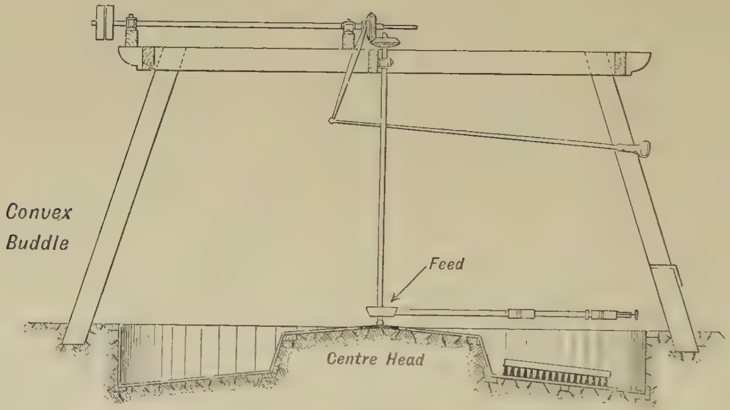


FIG. 247.

Convex Round Buddle.—Sectional Elevation (Holman Bros.). Shows sweep on radial arm, and the means whereby this sweep may be raised as the mineral bed collects (p. 341).

The concave buddle bears just the same relation to the convex as the concave revolving-table does to the convex table; the former produces a poor but bulky concentrate, the latter a concentrate which is rich but limited (Fig. 248). Both appear to have been used indifferently; the concave has this little difficulty, however, that the holes of its central discharge are not so conveniently placed for plugging. The tail-board, there, is a cast-iron ring about 2 ft. or so in diameter, provided with appropriate holes, the slime dropping to a launder passing out under the bed. With either type, when the collecting space is reasonably full, concentric rings are drawn to mark the boundaries of the different products, which then are shovelled out separately. Being of larger capacity than its rectangular prototype and sweeping being done mechanically, this appliance, in addition to its use upon enriched products, was also used to take the crushed pulp directly from the stamps. Treating such material, the head might be of sufficiently good grade after rebuddling on another machine, to be kieved; the middling would require two or three additional buddlings before any similar product was obtained; while the third product would probably be poor enough to discard.

Though these buddles even in Cornwall are no longer used to take the material directly from the stamps, they undoubtedly have a property of saving those fine mineral particles which in modern plants escape the shaking tables to which, amid a mass of larger particles, they are sent by classifiers. The buddle is, in addition, one of the few machines, which, while concentrating, effects sizing also, the coarse being found at the outside and the fine at the centre. It is the cost of handling the material which has made this appliance irrational under modern conditions. Its use, however, at a later stage in the operations, to enrich concentrate otherwise obtained, is still extensive in Cornwall and elsewhere; for such work, whether the material be coarse or fine, it is effective (Fig. 281).

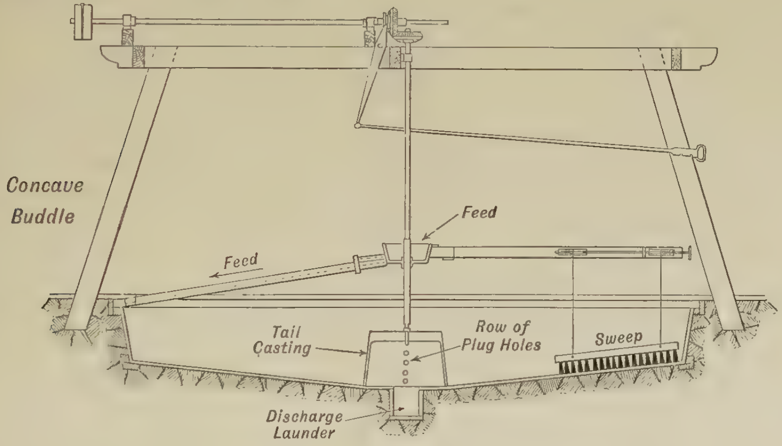


FIG. 248.

Concave Round Buddle.—Sectional Elevation (Holman Bros.). Shows radial feed arm and radial sweep arm (p. 342).

For cleaning granular concentrate, a streaming box is sometimes used wherein the feed enters at the upper end to be scraped up a curved side by knives set upon a framework revolving close over the bed, around a longitudinal axis. Pouring down this curved side, a stream of clear water drives the poor material down and away, while the knives secure a lateral progression of the concentrate. This is the Knife Buddle.

When pulp pours into a flat-lying box to settle as it will and without the aid of sweeps, a rough classification takes place, the coarse and rich material at the head, a finer middling in the middle, and a still finer and poorer tail. Such a flat settling box is known as a 'strip' (Fig. 246). In the absence of the effective stream which sweeping and a proper regulation of the overflow-level, secures, less enrichment is effected by such an appliance and the products are less regular. Strips were formerly employed in Cornwall just below the stamps. These were about 20 ft. long, 18 in. wide, and 15 in. deep, laid at an inclination of about 1 in 40, and with an overflow lip at the lower end. Worked in pairs, one filling, the other being emptied, they gave three sandy products for separate treatment, and a slime overflow.

Coming now to film-sizing over a riffled surface, such a surface may be impressed, for instance, upon glass sheets, and these may be laid on the ordinary inclined-table. Experiments made with such riffled or fluted glass-surfaces, with riffles about one-sixteenth of an inch deep and across, have shown that, when treating granular material—reground sand, for instance—they make a good recovery and effect a high enrichment in a very short length.[1] The gangue particles, which settle in the upper riffles, appear to be ousted by mineral-particles following; moreover, the riffling appears to delay the formation of banks and channels by causing the pulp to spread laterally. No advantage has, however, been found when treating the original slime, and such riffled glass-surfaces have the disadvantage of being expensive.

For coarser material, rectangular tables with a riffled surface fashioned in cement or sand-concrete, have sometimes been used; sometimes also long riffled cement-strakes have satisfactorily replaced canvas tables and blanket strakes; while, for treating alluvial material, especially that which has been screened, sluice tables with riffles or stops are standard practice. Standard also are the upstanding riffles set in sluice boxes treating coarser gravel.

Sluice Boxes and Sluice Tables.—Sluice boxes are open boxes laid in the transporting stream, a stream generally coming from the hydraulic exploitation of a gravel deposit. The ordinary length of a single box is about 12 ft. The total length of a sluice is made by assembling such unit lengths, 300—500 feet being a common figure; the necessary length increases with the fineness of the particles it is desired to recover, and with the velocity of the stream. Width and depth depend upon the volume and velocity of the stream, these factors, again, depending upon the amount and size of the material treated. Often, after the main sluice ends, the stream continues through a flume or ditch known as a 'tail race,' to carry the worthless debris away. Sometimes again, at the upper end the sluice is more carefully constructed to serve as a cleaning box. Ahead of this again, a 'head race' may sometimes bring the material to the sluice.

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[1] Truscott, *Trans. I.M.M.* Vol. XXVIII. p. 46.

Though, generally, the particles to be recovered are fine, the accompanying gravel is almost invariably coarse. Sluicing this mixed material with a given amount of water, the sluice boxes are designed that while the largest pieces roll, much of the remainder travels in suspension. Always provided that the depth of water is sufficient to submerge the largest piece, sluice boxes are wide rather than deep; width gives capacity. Ordinary dimensions for width and depth are 3 ft. and 2 ft. respectively, a portion of the depth being freeboard. But where the amount of material is extraordinarily large, the width may be as much as 10 or 12 ft.; exceptionally a width of 16 ft. and a depth of 4 ft. have been reached.

The material generally arriving unsized and incompletely cleaned, the rush of water must be sufficient to move the larger pieces and to break up any clay-bound lumps. The water velocity may thus be as much as, or even more than, 12 ft. per second, and though, in due course, the finer material would go below and the coarser material would ride above, it is probable that nothing but granular gold or very coarse mineral would settle behind riffles. Before the fine gold or mineral could be caught, the coarse material and the water necessary to move it, would have to be removed. This removal is effected by placing a screen in the bottom of the main sluice, through which screen the fine material drops for less vigorous treatment below, while the coarse material continues forward to the waste dump. Such a lower sluice is termed an 'undercurrent'; it consists of wider and shallower boxes or tables (Fig. 249).

Main sluices are laid at an inclination from $\frac{1}{4}$ in. to $\frac{3}{4}$ in. per foot, this low inclination giving the stream sufficient depth for complete submergence of the large pieces; depth in turn gives less wetted-perimeter, reduction of perimeter conducing to velocity. Low inclination is also often imposed by the contour of the ground; sluices treating coarse material have to be long if any adequate recovery of the fine particles is to be made, and a high inclination would therefore often demand more height than was available. High inclination also tends to pack the riffles.

For finer material greater inclination is demanded, because to be of full use the stream must be shallow and the frictional drag of the increased wetted-perimeter becomes a greater factor. Accordingly, with undercurrent tables, for instance, the inclination may be from $\frac{3}{4}$ in. to $1\frac{1}{4}$ in. per foot, this higher inclination, together with the greater width and the much smaller volume of water, making a much shallower and slower-moving stream. In such a stream, since the fine-mineral deposits early, a length of about 30 ft. generally suffices, though much longer tables are sometimes seen.

The above relation of low inclination for coarse material and higher inclination for fine material is, it will be remarked, the reverse of that obtaining in treatment on slime tables. The explanation of this reversal is, that in sluicing the necessities of transporting are paramount, those of separation receiving but second consideration.

With auriferous gravel it is the invariable rule to use riffles in the main sluice, and generally also on the undercurrent tables. Riffles, by the eddies they cause, keep the space immediately behind them from becoming packed with heavy sand, the gold settling in this free space. Such riffles are of three kinds: 'block riffles,' a pavement of stout wooden blocks placed an inch or so apart, in rows spaced wide apart; 'pole and bar riffles,' these being lengths respectively of lumber and steel held in head-boards and placed longitudinally about an inch apart in the bottom; and angle-iron or 'Hungarian riffles,' these being placed transversely with the solid angle uppermost and against the stream, about two inches apart (Fig. 246).

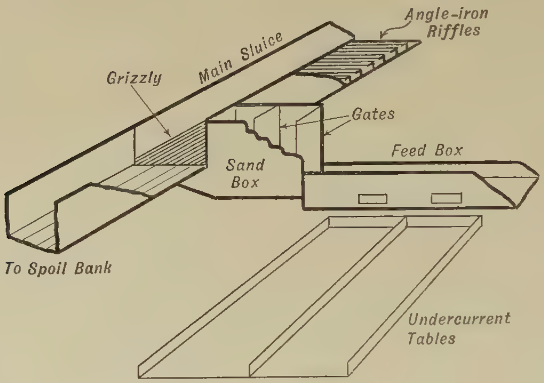


Fig. 249.

Main Sluice and Undercurrent.—Perspective Diagram. This illustration indicates the main sluice where coarse and granular gold is caught behind riffles under a strong and deep stream in which large boulders are transported. It shows also the grizzly laid in the bottom of the sluice to provide opportunity for the fine sand to drop out of the stream. This sand drops into a box provided with gates open to such an extent that, while permitting a continuous outflow of sand at the bottom, they maintain a high level of water within. In this way the bulk of the water remains in the main sluice to carry the large material forward to the spoil bank.

The feed box into which the sand is delivered runs along the head of a proper number of tables, distributing the sand to them all. In the diagram only two tables are indicated, but there might well be more (p. 345). (Ulrich, *M. & S.P.*, June 5, 1915, p. 867.)

Wooden riffles have the disadvantage that they must be nailed in place or otherwise they might float ; being nailed they are not so readily removed at times of clean-up, for which reason block riffles sometimes give way to a pavement of rock boulders. On the undercurrent tables the riffles are less deep, angle-iron of less than one inch in dimension being common ; or sections of expanded metal laid on matting ; or burlap roughened by chicken wire.

The sluices of great length and low inclination, of relatively great depth and high velocity of water, the sluices designed to catch the gold by a long riffled bottom while the coarse material is transported above, are sometimes termed Alaskan sluices. The Russian sluice, on the other hand, is a broad shallow sluice of greater inclination and shorter length, designed to catch the gold in the first ten or twenty feet ; like the undercurrent table which it resembles, it only treats screened and cleaned material.

In sluicing stanniferous gravel, ordinary riffles are not employed. Tin concentrate is much the same class of material as the heavy sand which the riffles of gold sluices ‘ boil ’ away ; moreover, the space available for collection behind riffles is too limited for the amount of this concentrate. Tin concentrate is caught in a bed of sand. Instead of riffles, stout battens are placed in the boxes at regular intervals of, say, 8 ft. or so, as ‘ stops ’ behind which the sand bed collects (Fig. 246). At first this bed will contain little mineral, but as sluicing continues, the amount increases, the descent of this mineral into the bed being promoted by digging and working the surface. Then, when mineral is seen escaping over a stop, another batten is placed upon that stop to raise the level, and so on until many tons of roughly-enriched material are collected and the sluice needs to be cleaned.

Nor are normal undercurrents used when hydraulicking stanniferous gravels. It being impossible to hold and collect tin concentrate behind riffles, recovery cannot begin in a stream strong enough to move large boulders, such boulders must first be removed. The material treated in the main sluice is, accordingly, relatively small ; moreover, the surface of the sand bed in that sluice is particularly good for arresting fine mineral. With the recovery of the mineral thus provided for, any necessity for a second treatment in an undercurrent is avoided by employing a proper length of sluice ; the sluice being wide and the stream not strong, a length of about 150 ft. is usually sufficient.

In cleaning-up a gold sluice the riffles are lifted, permitting the concentrate either to be shovelled out for treatment in a better prepared sluice box, or, as is more usual, to be further concentrated in the main sluice itself, under a flow of clear water. If mercury has been used behind the riffles, this is carefully collected and strained to recover the amalgam. The clean-up of a tin sluice is effected by raking the collected sand upwards against a stream of clear water, the waste being carried away by removing first one stop, then a second, and so on, till the bulk has become so reduced and the concentrate so enriched, that further work is best conducted in a special tye or cleaning box.

Clean-up takes place after the regular intervals which experience has shown sufficient for a proper accumulation of valuable material. Since this removal from the whole length of a sluice takes considerable time, a part clean-up of the upper boxes is more often made, say at intervals of a week, leaving the more extended work of a complete clean-up to be undertaken at intervals of a month, six months, or a season. Sluice tables being shorter and having less collecting capacity, are sometimes cleaned-up each day.

Gold and tin gravels, in addition to exploitation by hydraulicking, are often won by dredging. Where such dredges include complete screening appliances, as most bucket dredges do, the undersize material is generally smaller than half an inch and perfectly clean. Such material is treated on sluice tables, those for gold being covered with suitably small riffles, those for tin being provided with the necessary stops; these tables have commonly a length of about 70 feet. On gold dredges it is usual to provide about 0·66 sq. ft. of table area per cubic yard of gravel dredged per day. Since about half the deposit is discarded as over-size, the table area in respect to the material actually passed over the tables is roughly double that figure. To catch the heavy black sand which escapes these tables, Neill sluice-jigs have been proposed and used (Fig. 234). On tin dredges there is a similar but somewhat more generous provision; the tin in dredged material is fine, little remaining on a 5-mesh screen, most passing 30 mesh. To catch such fine mineral it is usual to allow 1·0—1·5 sq. ft. of table area per cubic yard of gravel dredged per day. This table area, whether for gold or tin, is for greater convenience spread over parallel tables, each with a width of about 4 ft.

The amount of the water which flows over sluice tables is considerable, and roughly about 15 tons per ton of material. This large amount, necessary in the first instance for transportation, seriously affects the recovery of fine mineral. On tin dredges, for instance, not much tin finer than 80 mesh is caught, though probably 30 per cent of the tin in the gravel is smaller than that. In sluice boxes, since larger material is transported, the amount of flowing water is much greater.

Résumé.—Reviewing this description of the streaming or film-sizing appliances, it will be realized that though the primary division was between appliances with a plain surface and those with a riffled surface, in respect to their shape these appliances may also be divided into: rectangular tables, from off which the concentrate is removed intermittently; round tables, which deliver concentrate continuously, bringing it to a particular point at the periphery; and belt machines which carry the deposited concentrate up the incline, against the descending streams of feed and wash-water, up over the end, to be detached as the belt dips into water. With belt machines, the wash-water adds its bulk to the feed stream, and adjustments of its amount cannot be made without affecting the settlement taking place below; with round machines the wash-water flows down a separate course and its adjustments in no way interfere with the separation being effected by the feed water.

Oscillating Concentrators, Vanners, etc.

If, during deposition of the mineral upon a flatly-inclined plane, that plane be oscillated regularly, a stratifying action is superimposed upon the ordinary stream-sizing action. One of the necessities with the appliances in the class previously described was the prevention of channelling or guttering of the surface. On those with a relatively smooth surface this channelling was prevented or minimized by allowing only the thinnest layer of mineral—theoretically only one particle deep—to be deposited before removal. Where, however, the settled material was allowed to collect, as in buddles and some sluices, the surface was kept even either by sweeps—as with the buddle—or by horizontal stops and by raking—as with the sluices and other streaming boxes. All this attention is avoided if the appliance be gently oscillated or shaken with a harmonic, that is, a non-differential, movement, this movement sufficing of itself to keep the surface regular and even. At the same time another advantage arises: the relatively high inclination which with the ordinary streaming appliances was necessary that the stream might take a straight and even course down the plane, is no longer necessary; where an inclination of $1\frac{1}{2}$ in. to the foot would be given with such appliances, oscillating or vanning machines would demand only half an inch. At this lower inclination, the velocity of the stream is appreciably less and its depth correspondingly greater. In that greater depth the oscillating movement keeps the particles loose, a condition favouring stratification according to density, and making possible a feed of thicker consistency. Under these conditions, also, there is a vertical sizing-effect, the finer particles finding less interference to their downward passage, the coarser particles forced to a riding position on top.

Obviously, it would not be possible to impart the described movement to heavy machines, or to those in which a good deal of material collects. Nor would it be economically rational to apply such a movement to machines whose great recommendations were their simple construction and low operating-cost. The great bulk of these oscillating machines are accordingly highly-developed machines. They are almost exclusively of the belt type, the belt being of carefully-prepared rubber, a material presenting a smooth, and at the same time a durable and impervious surface.

Before describing these belt machines, popularly known as Vanners, the order of the previous sequence is maintained by taking first, infrequent though it be, the application of this oscillating movement to rectangular and to round machines. Occasionally, for instance, blanket strakes, to recover unamalgamable coarse gold, are placed close under the stamps and ahead of other possible concentrating appliances. Usually, in that position no great space is available and only a short blanket strake can be inserted ; a side oscillation or shake makes that length much more effective. At the Ouro Petro mine, Brazil, such blanket strakes, only 4 ft. in length, recover about 70 per cent of the gold content of the ore crushed. Exceptionally also shaking sluices have been used. The application, however, of this movement to rectangular inclined-planes is on the whole quite unimportant. With round machines the position is very similar ; such a movement imparted to a rotating table is, on the face of it, not so readily accommodated. One such appliance, however, is worth mention ; the Sperry table is a round table about 15 ft. in diameter, its convex bed being sloped about $1\frac{1}{8}$ in. per foot. In the ordinary way it rotates about once every two minutes, while in addition, the vertical spindle of the machine, and the whole bed with it, is gyrated through a very small circle of about $\frac{1}{4}$ in. diameter, the number of such gyrations being about 300 per minute. The effect of these gyrations is undoubtedly to promote stratification; the movement is not sufficient to bring into suspension again any material already settled, but on the contrary it promotes the stratification of mineral below and gangue above. With such a gentle movement, this table is suitable for the treatment of slime ; it is, indeed, not a sand table.

The Frue Vanner.—The outstanding machines of this class, however, are the belt machines, and of these, that known as the Frue vanner is the representative and prior type. In this machine, the inclined plane is formed by an endless rubber belt, normally six feet in width and with flanges at the side, which revolves round two horizontal rollers each about one foot diameter, the upper of the two being the driving roller (Fig. 250). These two end-rollers are about twelve feet apart, and capable of being so set that the lower one may be as much as six inches lower than the upper. The belt being pulled up the incline so formed turns over the upper roller, not to return immediately, but before doing so to pass around two guide-rollers, the first of which takes it down into a water trough, the second in turn lifting it from the trough to pass around the lower end-roller. The two end-rollers are directly on the ends of a frame, upon which small subsidiary rollers spaced about a foot apart support the stretch of belt which forms the inclined plane, and keep it from sagging. This frame, in turn, is carried upon six steel blades or springs upstanding from the main frame below (Fig. 251). Supported on this main frame, also, at one side, is the driving shaft from which the oscillation and the gradual travel of the belt are effected. The oscillation is given by three cranks, one at each end and another in the centre. The belt travel is effected through cone pulleys driving a worm and a worm-wheel, this latter being co-axial with the upper roller, to which roller the revolution of the worm-wheel is conveyed by a spiral-spring coupling.

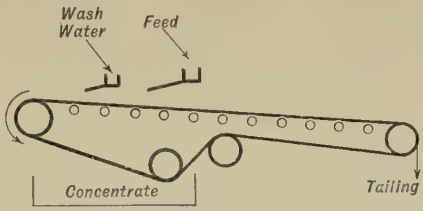


FIG. 250.

Frue Vanner.—Diagram. The arrow round the upper end-roller indicates the direction of the belt travel. Slope of bed about one in forty (p. 350).

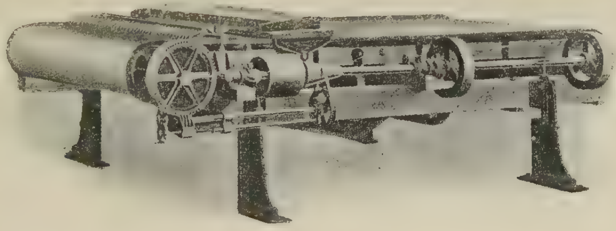


FIG. 251.

Frue Vanner.—General View (Allis Chalmers). This view shows the driving side, with driving shaft, wide driving-pulley, cone pulley for belt travel, and heavy fly-wheels. The three eccentrics or cranks, by which the belt is oscillated, are situated one at each end of this driving shaft and one near the central fly-wheel. The worm drive of the upper roller and the adjustment of the speed of that drive by altering the position of the small belt upon the cone pulley, are clearly indicated. Three upstanding springs supporting the wooden belt-frame upon the main frame are also seen; apparently there are five on either side (p. 351).

The cone pulleys just mentioned allow the belt travel to be adjusted from a minimum of about two feet per minute to a maximum of about six feet. The amplitude of the oscillation is fixed by the crank-throw; for slimy material this amplitude would be about one inch, for sandy material an inch and a half or even as much as two inches. The rate of oscillation varies from about 185 to 210 per minute, the smaller number going with the larger amplitude. The slope of the bed, as already stated, can be as much as six inches along its length, this being equivalent to half an inch per foot, from which maximum it can be brought to practical horizontality; an average inclination is about 1 in 40.

On to the plane so formed and supported, the pulp to be treated, having a consistency of about 4 of water to 1 of ore, passes from a distributing board, situated about three feet from the upper end and facing upstream. From this board the drop to the belt is about an inch and a half; the depth of stream upon the belt is about a quarter of an inch. Above the pulp distributor comes a row of wash-water spouts also facing upstream. Through these water-jets, the concentrate which has travelled with the belt up past the point of the pulp arrival, continues as parallel streaks round the end roller, eventually to be washed-off into the concentrate box (Fig. 252). At the other end of the machine the tailing flows into an appropriate launder.

The Frue vanner is particularly suited to treating material smaller than 30 or 40 mesh. From such material, slime included, it makes a satisfactory recovery. The sandy portion conceivably prevents the too rapid passage of the slime to the overflow, giving it protection and opportunity to get down to the belt. Once settled upon the belt no further movement of the mineral is necessary, the belt travel providing all the necessary progression; the tighter the mineral clings to the surface the more surely is it saved. This is the principle upon which all slime concentrators act, the revolving table, for instance; it will be seen later that, with machines which are essentially sand machines, the mineral itself, by virtue of momentum received, secures its own progression. Treating mixed material below 40 mesh, a vanner six feet in width would have a capacity of about six tons per day of 24 hours, or, one ton per foot of width per day. With slime, say the material passing 200 mesh, the capacity would be less; with sandy material from which the slime had been removed, it would be greater.

Capacity is also affected by the proportion of mineral in the material treated; where this proportion is higher, capacity is greater, there is less gangue to be removed. Making no middling product, the particular province of this vanner is the treatment of material which contains but a small proportion of mineral, say something less than 5 per cent, and the above figures of capacity are in terms of such material. The Frue vanner first established itself by the satisfactory recoveries it made of auriferous pyrite from crushed ore, after this ore had passed the amalgamating plates of a stamp battery; from such material it separated the greatest portion of the valuable pyrite, leaving the tailing in an impoverished and worthless condition. Vanner concentrate is, generally, not very clean, chiefly because, in the absence of a middling product, loss in the tailing is thereby avoided, but also because of difficulty lying in the fineness of the material treated. With many ores, copper ores and auriferous pyrite for instance, the subsequent metallurgical treatment does not demand a high-grade concentrate.

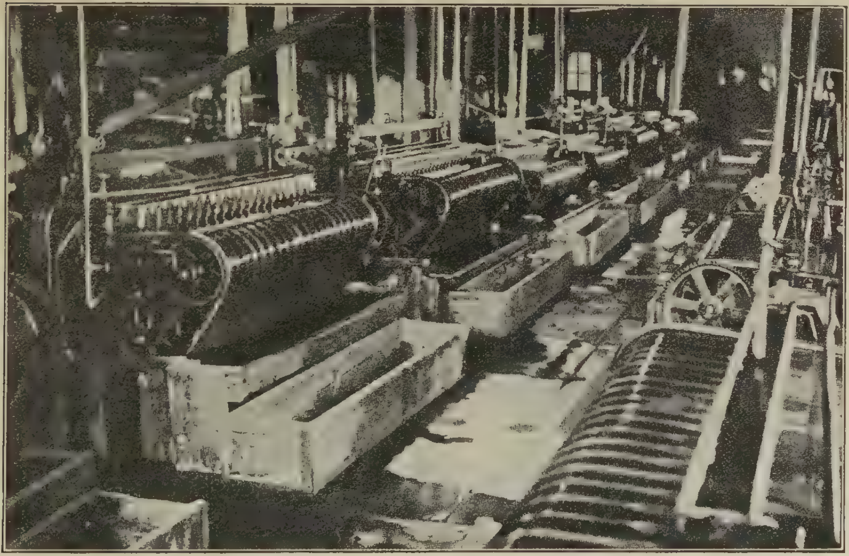


Fig. 252.

Johnston Vanner.—View of Vanner Floor. The streaks formed as the concentrate passes through the wash-water jets, are clearly seen (p. 352).

Though early introduced the Frue vanner is still in extensive use, hundreds of them working in some dressing mills. Each machine takes about half a horse-power; about four tons of wash-water are required per ton of ore treated; a belt generally lasts about two and a half years. They require considerable attention; namely, to see that the distributors are working freely and the belt running truly; to renew subsidiary rollers, and take up belt-stretch, etc. These points, while they do not seriously detract from the many good qualities of the vanner, make it somewhat expensive in operation and in repair.

Minor modifications in design have effected minor improvements. The belt of the Frue vanner, being supported on upright springs, is at its highest position at the middle of the stroke. Swinging on these springs

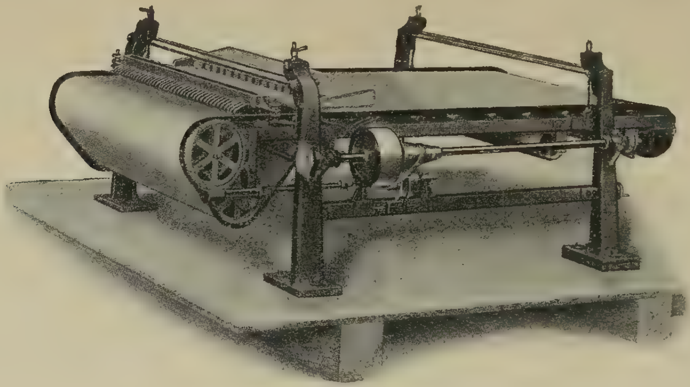


Fig. 253.

Johnston Vanner.—General View (Allis Chalmers). Here the steel belt-frame is suspended from four standards, two on each side. Oscillation is effected by two eccentrics, one at each end of the driving shaft (p. 355).

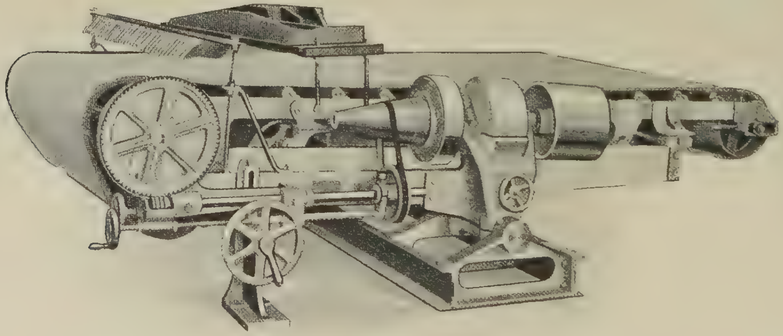


Fig. 254.

Isbell Vanner.—General View. The steel belt-frame of this vanner is supported upon the ends of large blade-springs set on edge, these blade-springs in turn being carried in central chairs fixed upon a stout central shaft. Upon that stout shaft all the working mechanism is borne, and about it the whole vanner may be given any inclination endwise. The wheel and standard near the head of the machine are the means whereby a particular inclination is given; the oscillation is horizontal, as the ends of the blade-springs can only swing horizontally; it is given by one single eccentric placed centrally, and here shown enclosed. Being a double eccentric the throw can be adjusted. The bearings of the lower end-roller are movable in a slot, this arrangement permitting any belt-stretch to be taken up (p. 356).

to right and left, it is conceivably a little lower at the ends of the stroke, and there is a tendency for the material to be thrown to the sides, where dry banks sometimes form. To remedy this, a type of vanner, the Johnston, suspends its belt from above, the belt hanging and swinging from four standards (Fig. 253). With this disposition the belt is lowest at the centre of the stroke and highest at the ends, the tendency being to bring the

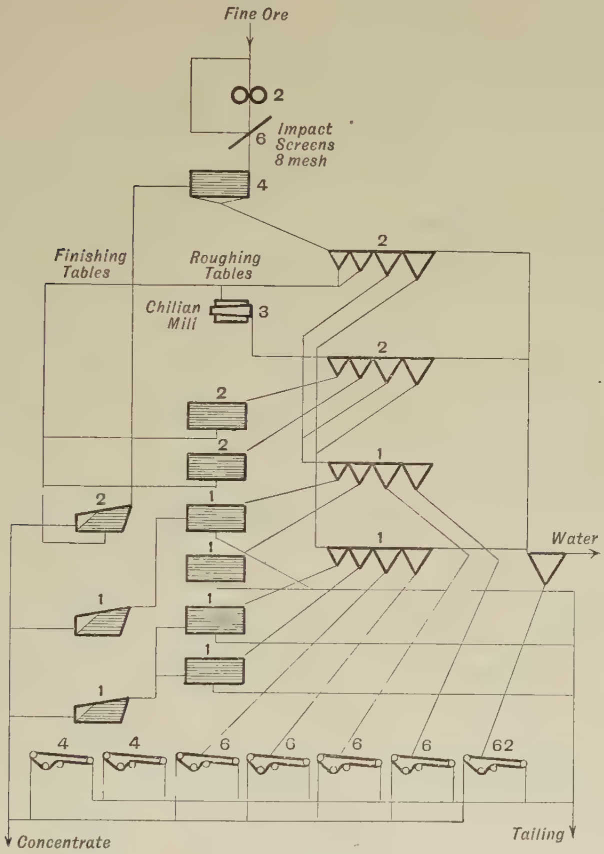


Fig. 255.

Water Concentration, Utah Copper in 1910.—Flow-sheet. The block figures represent the numbers of the respective machines in a Mill unit having a capacity of 1000 tons per day. The Roughing tables have the Wilfley mechanism, but the riffles go right across; it will be noticed that they make no finished product, neither concentrate nor tailing. The Finishing tables are normal Wilfley tables. The scheme of treatment here outlined represented a considerable advance on previous methods of grinding all the ore for treatment on finishing tables. It has, however, since been radically altered by the introduction of flotation (pp. 357, 376, 379, 647).

material to the centre. In the Isbell vanner, again, the movement is made entirely flat, the belt swinging upon four horizontal springs only capable of moving in the horizontal plane (Fig. 254). This Isbell vanner differs also from the general design in that the whole machine is supported

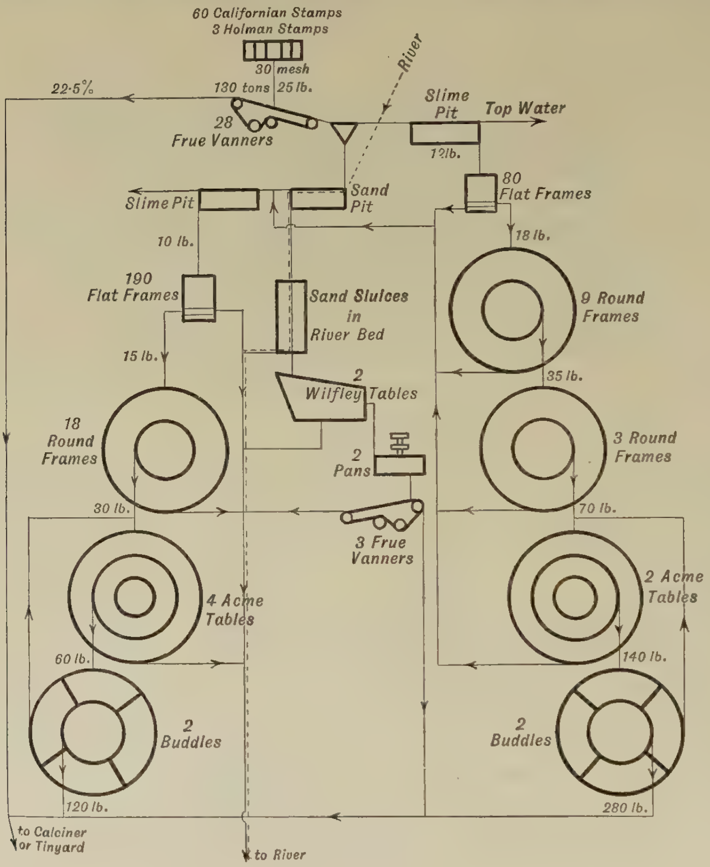


FIG. 256.

Water Concentration, Dolcoath, Cornwall, 1918.—Ore contains fine-grained cassiterite, but neither wolframite nor mispickel. The Acme table is a double table, a convex table inside and a concave table outside, the latter making a concentrate which is treated by the former; the beds of both tables are short. The distinctive feature of this milling scheme is that the whole stamp-pulp goes unclassified to vanners. The rate at which the slime concentrate is enriched is very slow, several treatments being given before that concentrate is rich enough to bear the expense of the treatment in the tin-yard; the slime tailing receives but one retreatment. The assays of the products are given in pounds of metallic tin per ton (pp. 357, 379, 647). (Truscott, *Trans. I.M.M.* Vol. XXVIII., 1919, p. 72.)

not upon an ordinary rectangular framework, but upon a stiff shaft at the centre, round which shaft the belt may be tilted and given any inclination. To allow this design without detracting from the necessary stiffness, this vanner has a shorter length of 10 ft. in the place of the more normal 12 ft. Another feature is that the movement is made by an eccentric at the centre, the stroke of which is adjustable; provided the vanner be not too long, a single drive of this sort conceivably gives a more perfect oscillation than the three cranks of the standard vanner. The number of oscillations made by these more modern vanners is lower than with the original Frue vanner, the Johnston vanner sometimes making only 140 oscillations per minute.

All the vanners so far described have been end-inclined and side-shaken. The Triumph and the Embrey vanners, while end-inclined, have likewise an end-shake, a design which increases their capacity for sand but disturbs the settlement of slime mineral. In another early type a circular oscillation was given, but this again was found to make the bed too shallow. The Senn vanner, an endless rubber belt 6 ft. in width supported over two end-rollers, 8 ft. apart, has a combined side and end oscillation, the frame being supported on ball bearings, discharge being at the end. Finally, the Craven vanner is distinct in that, being end-shaken, it is side-inclined, delivering its products, which may include a middling, over the lower side.

In the place of the ordinary plain belt, a belt with about eight corrugations to the inch is sometimes used. Such a corrugated belt appears satisfactorily to treat material from which the slime has been removed, but with mixed material it gives a poor concentrate.

Résumé.—Though at times, as in the recovery of auriferous pyrite, the vanner may treat all the ore and be the only concentrator, in the treatment of base-metal ore it is more usual to find that only the finest sand and slime are sent to it (Fig. 255). Exceptionally, with stamp-crushed tin ore, vanners may be used primarily as sand machines, though even then it will probably be found that the whole pulp from the stamps, slime included, is treated by them (Fig. 256).

That vanners have a distinctive movement is shown by the fact that if a small coin be placed upon a belt in operation, good metal though it be, it will find its way into the tailing and not into the concentrate.

Differential-stroke Vanners.—With the normal oscillating vanners it is convenient to describe one or two similar appliances, which, though they have a differential stroke, make this stroke only to stratify the material and not with the intention of jerking the denser mineral forward. Such an appliance is the Luhrig vanner, which consists of an endless rubber belt unflanged, running round two end-rollers, and side-inclined. The inclined bed so formed, moving over a water-lubricated wooden platform and not upon subsidiary rollers, is about 10 ft. long and 4 ft. wide (Fig. 257). Fed at the near upper corner, and the belt travelling forward at an ordinary rate of 6—8 ft. per minute, the settled material goes forward, each particle along a diagonal line, the resultant of the belt travel and of a film-sizing movement downstream under the action of wash-water delivered along the upper edge. With a properly-partitioned launder inserted under the lower edge, the products are taken off in the order, tailing first, middling second, and concentrate last (Fig. 258). The differential stroke is given by the revolution of a many-cammed wheel generally at the forward end, the cams pressing the vanner back against a spring which in turn gives a quick forward motion ending against a stop. The resulting percussion promotes stratification much in the same way as the knock on the side of a kieve.

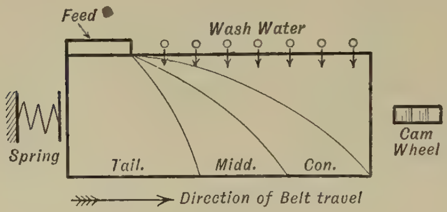


FIG. 257.

Luhrig Vanner.—Diagram illustrating manner of separation (p. 358).

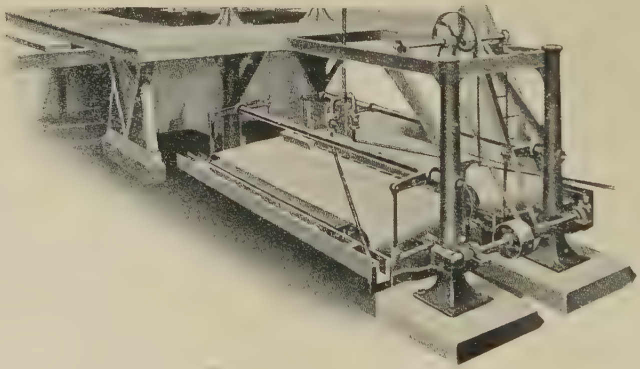


FIG. 258.

Luhrig Vanner.—General View. A duplex arrangement of two vanners is here shown. Supported on iron-work, the side inclination of the belt is clearly seen, while the possibilities of changing this inclination are evident. The feed-box is shown at the rear upper corner; and, further behind, the spitzkasten in which this feed is thickened. In the foreground is the driving mechanism, the cam wheel being seen. At the side is the launder properly divided to collect the different products. The wash-water arrives through a pipe diagonally disposed along the bed (p. 358).

Similar machines are the Monell vanner, used in Colorado, and the Weir-Meredith vanner, adopted at Broken Hill. The 'head motion' of this latter is of the single-toggle type—that of the Wilfley table presently to be described has double toggles—while the belt is supported upon a wooden platform or water-bed, this method of support giving a truer surface than one supported at intervals upon rollers. Another similar end-shaken vanner is one having its frame supported upon upright wooden springs, while the end movement is made by an eccentric.

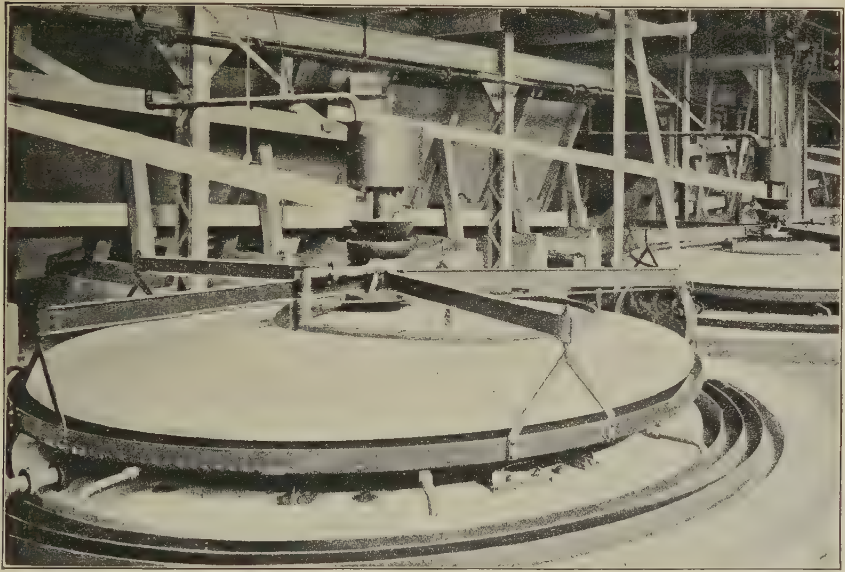


Fig. 259.

Bartsch Table.—General View. Except for a small percussive movement, the bed of this table is stationary, the different products being separately collected by rotating the collecting launders. These collecting launders are sections of a complete annular launder close up under the table periphery. The tailing section comes first to catch the tailing which streams quickly down, pouring this tailing into one of three outside annular launders, whence it passes away. Then comes the middling section, which by a discharge pipe of proper length leads its product to a second annular launder; and so with the concentrate. The feed and wash-water services also rotate; the latter is a spirally disposed pipe over the bed (p. 359).

Differential-stroke Round-table.—As previously mentioned, the Sperry table whilst revolving regularly round its vertical axis, makes a number of extremely small gyrations, this gyratory movement assisting in stratification and settlement. Another round table, the Bartsch, is given a small percussive stroke with the same intention, and accordingly may be considered to be among the appliances now being described. The Bartsch table, however, like the Linkenbach table, is stationary, the separation effected being consequent upon the revolution of the feed and distributing services (Fig. 259).

JERKING CONCENTRATORS; TABLES, ETC.

If, while the particles of pulp are moving down an inclined plane under streaming action, that plane be given a jerk or bump at the side, at each jerk the particles will move across the stream by reason of the momentum communicated to them, the denser particle farther than the less dense, the larger particle farther than the smaller. Furthermore, those particles which travel slowest down the plane, receiving a greater number of impulses before leaving it, move farthest across. Accordingly, the combined result of these two movements down and across the plane, will be that the denser particle follows a flatter diagonal than the less dense, and that, of particles having the same density, the smaller takes a flatter diagonal than the larger. With this result once established, then along any horizontal line across the stream the fine mineral will be found in advance, the large mineral associated with the fine gangue will follow, while the large gangue will be in retard (Fig. 260). This same arrangement will also obtain down the stream, the transverse movement not upsetting the normal streaming arrangement.

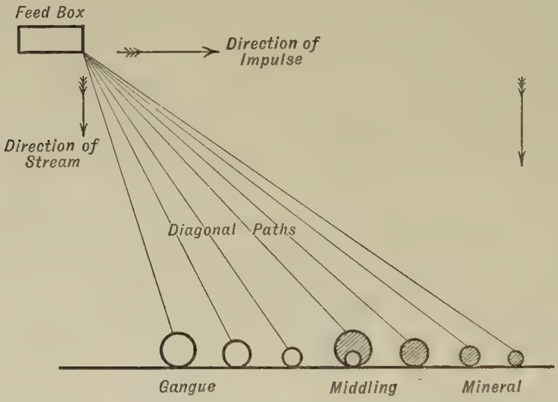


FIG. 260.

Jerking Tables.—Plan of Separation. The middling is shown as having a counter-classified character (p. 360). See also Fig. 261.

The movement given to the table to effect this transverse separation of mineral and gangue must be differential, that is to say, the forward and backward strokes must differ; the former, for instance, may begin slowly and end abruptly, the backward stroke accelerating in the reverse direction; or there must be some equivalent difference. Beginning slowly, the material resting on the table is carried forward with it; ending abruptly against a stop or upon quick reversal of direction, the material continues forward.

Rittinger Percussion Table.—The earliest appliance of this class was the Rittinger table, a smooth wooden plane, say 4 ft. wide and 6 ft. long, hung by rods from four standards—two on each side—in such a way that, after being pushed to one side by a cam, a spring compressed by that movement brings the table smartly back against a stop, gravity assisting (Fig. 261). The inclination of this table was about 1 in 20, the number of complete movements about 120 per minute, and the amplitude of the movement about 1 inch.

Fed at the rear upper corner the particles take a diagonal course down the plane, the denser particles moving farther across the table, partly because they move slower down the plane and consequently receive a greater number of impulses before they leave it, and partly because they are thrown farther at each impulse. The gangue, on the other hand, feeling the external impulse less and the streaming action more, takes a more direct line downstream. At the bottom, a suitably-partitioned launder collects, and allows to be led away, the products so separated, these generally including a middling. For the more precise division of these products and their deflection into their proper partition, adjustable fingers are fixed upon the lower edge of the table.

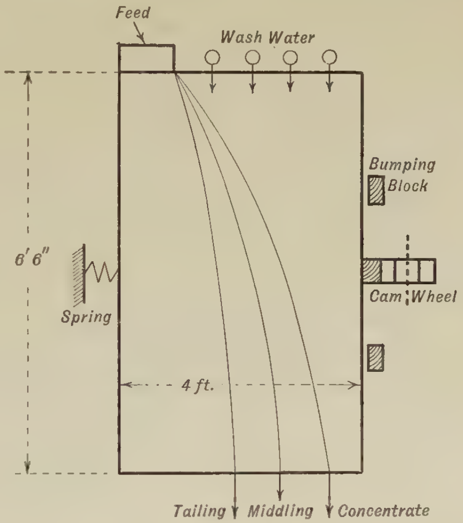


Fig. 261.

Rittinger Table.—Diagram of Separation. The paths taken by the different products are not straight diagonals but in their upper portions roughly parabolas, the velocity of the wash-water being slow at the beginning (p. 361).

Discharge, it will be realized, is effected by the stream, while separation is brought about by the external impulse. The water necessary to discharge the tailing, in so far as that takes place below the feed-box, is that entering with the feed. Away from that line discharge is effected by added water, of which there is a service across the head of the table. It is added water, also, which discharges the middling and the concentrate in succession, the latter in passing farther through this water being suitably cleaned. The amount of water coming with the ore is generally about 10 tons per ton of ore, while the added or wash water is about 20 tons. This great amount of additional water, and low capacity, militated against the wide adoption of this table. It undoubtedly made a very clean separation of mineral from gangue, and, when two or more minerals were present, of mineral from mineral. In modern practice, however, it has no place.

Wilfley Table.—Employing the same idea of a jerking motion across the stream, the Wilfley table improved the performance by laying riffles in the direction of the motion. In the bed formed between these riffles stratification takes place, the mineral sinking to the bottom, where it not only lies withdrawn from the streaming action but feels more fully the external impulse; the gangue, on the other hand, being raised into the stream, is carried away. With this positive retardation to the descent of the mineral, a relatively short length downstream suffices even for a rapid rate of feed; on the other hand, the more effective impulse given to the mineral permits a greater dimension across the stream, this greater dimension giving greater opportunity for the mineral to move clear of the gangue. The net result of the presence of such riffles is that treatment is quickened and capacity increased. In addition, the concentrate being moved to its discharge by the external impulse and not by wash-water, less of the latter is required.

Actually, the Wilfley table is 16 ft. in length across the stream, 6 ft. in width downstream at the feed end and 5 ft. at the forward end; only roughly is it rectangular, the diagonal followed by the pulp being longer than that which the pulp crosses (Fig. 262). The surface is of linoleum; the riffles nailed upon it are of wood. Of these riffles, that at the bottom extends the full length of the table, being about half an inch deep at the feed end and the depth of a feather edge at the other end. Up the table, the remaining riffles, all beginning at the feed end, extend less and less toward the concentrate end, till the top riffle is only about 4 ft. 6 in. in length; the line joining their forward ends accordingly cuts diagonally down the table, this disposition being one of the features of the appliance. The upper riffles are also less deep at the feed end; all taper to a feather edge. Spaced about an inch apart and being themselves half an inch wide, there are in all about 45 such riffles.

The riffled plane so constituted is supported on carefully selected boards laid parallel to the longer diagonal, these boards resting upon longitudinal runners, the whole constituting what is described as the 'deck.' This deck in turn is supported upon two rocking plates disposed at right angles to the motion, one near either end. These plates sit upon steel chairs, which can be tilted around the longitudinal axis of the table to give any desired inclination downstream. These chairs in turn are upon the main frame (Fig. 263).

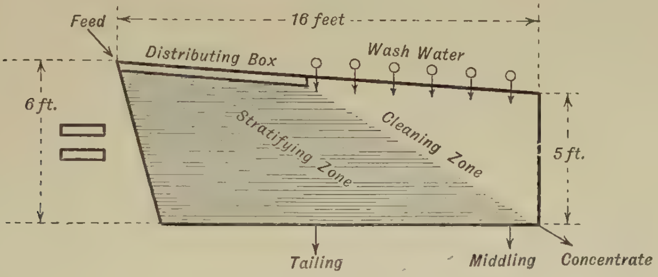


Fig. 262.

Wilfley Table.—Diagrammatic Plan. Left-hand Machine (pp. 362, 365).

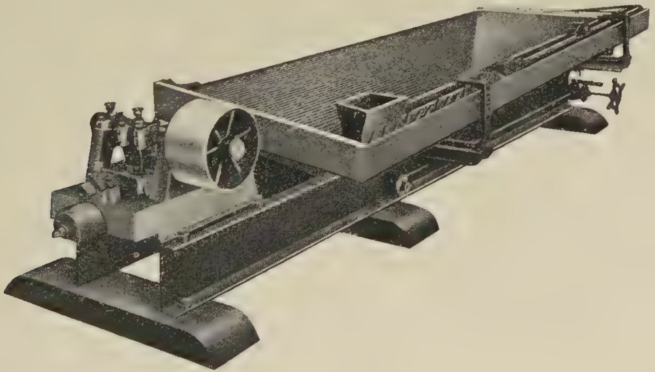


Fig. 263.

Wilfley Table.—General View from behind. This illustration shows the fixed back-guide of the head-motion and the adjusting screw, the feed-box and distribution, the water service, and the tilting gear. Running along the concentrate end will be noticed the drip-bar which brings water from the water-box to keep that edge of the table wet. The feed-box is fixed to the deck, the water-box to the immovable frame (p. 363).

At the rear upper corner, and extending the length of the upper riffle—that is, for about 4 ft. 6 in.—is the feed distributing-box, which is attached to the deck and moves with it. In line with this box, but separate from it and fixed to the floor, is the wash-water distribution, which extends to the forward end of the deck, and thence is carried down the concentrate side. Along the bottom are the discharge chutes, that for the concentrate being normally at the forward corner, the middling coming next, and then the tailing in the rear.

The mechanism or ‘head motion’ by which the jerk is effected is very similar to that which brings about the reciprocating movement of the jaw breaker (Fig. 19). There is the same long eccentric upon a driving shaft; the same pitman and toggle-plates—these plates, however, being inclined upwards instead of downwards from the pitman; the same relation

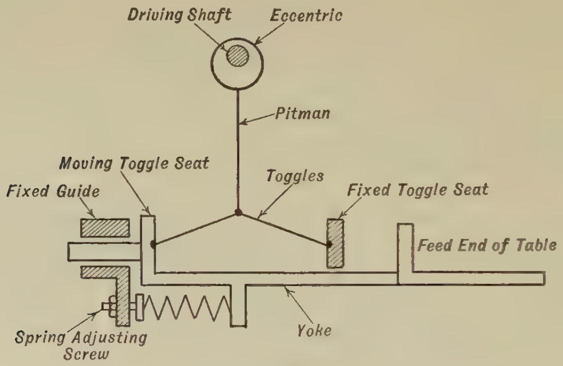


FIG. 264.

Wilfley Table.—Diagram of Toggle Mechanism. Actually, the yoke does not pass below the fixed toggle-seat but around it. The strength of the spring can be adjusted by the screw indicated. The length of the stroke is adjustable by raising or lowering the fixed toggle-seat (p. 364).

of the toggle seats, the one immovable, the other receiving all the spread (Fig. 264). With the descent of the pitman the rate of spread is relatively great at the beginning, and slow at the end. Such a movement being the reverse of what is required for the forward stroke, by fixing the forward seat the spread of the toggles is made to accomplish the backward stroke. To that end the movement of the rear seat is conveyed to the table by a yoke which passes forward and beyond the fixed seat. The toggles, while capable of forcing the seats apart, are not capable of bringing them together again, but in their spread, a spring is compressed against the fixed frame of the machine, energy being thereby stored to make the forward stroke. In accomplishing this stroke, the spring does not fly out, but extends under control of the toggles. Accordingly, except that the spring takes up any looseness present, and may somewhat hurry the forward stroke by assisting the pitman upwards, the only difference between the two strokes is that the forward stroke increases in velocity to the end, the backward stroke decreasing. At the forward end, in consequence, reversal of direction is smart, while at the backward end it is relatively slow. A graph of this complete movement, drawn to space as ordinates and time as abscissae, shows a curve with a relatively pointed maximum and a rounded minimum, this curve being symmetrical with respect to the ordinates passing through these extreme points (Fig. 265).

The rate of reciprocation is about 240 per minute for sandy material and somewhat higher for very fine material, the amplitude varying from 1 in. to $\frac{1}{2}$ in. respectively. Adjustment of the rate is made from pulleys on the line shaft from which the power is taken ; adjustment of the throw can be made by raising the fixed seat which, making one toggle flatter, diminishes the spread.

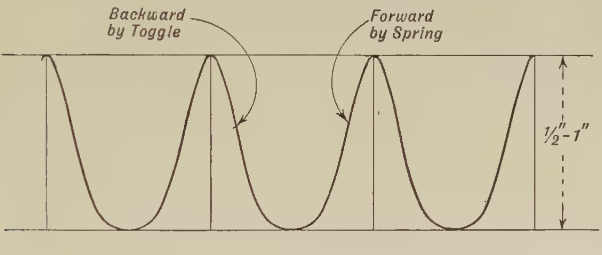


Fig. 265.

Wilfley Table.—Graph of Stroke. The forward stroke, though made by the spring, is controlled by the toggles (p. 365).

The working of the table is as follows : the feed, arriving in the feed-box at the rear end, is distributed over four feet of the table, that is, along the length of the shortest riffle. Becoming stratified between the riffles, the mineral particles, forcing their way below, come into contact with the table surface, in which position they feel every impulse. Thrown by these impulses along the riffle these particles gradually lose the protection of its depth, till at last they come out into the open cleaning area (Fig. 262) ; the larger particles, indeed, may not arrive so far, but may have been washed downstream already. Dropping to the next riffle, the mineral is cleaned by the wash-water ; then, carried forward by that riffle till the same thing happens again, this stepped progression continuing till the far corner of the table is reached and the now clean concentrate is discharged. Meanwhile the gangue particles riding on top are washed more directly downstream, into and then out of each bed in succession, till with little delay they are discharged.

It will be realized from the foregoing that the portion of the table in most active operation is a diagonal zone following the riffle ends. The upper portion beyond this zone and the lower portion below the feed-box are more or less out of action, for which reason the table is shaped with diminished extent in these two directions. The crowding of the work into this one zone somewhat limits the capacity of the machine, a capacity which is ordinarily about 20 tons per day when treating sand from 5—30 mesh, and about half that tonnage with finer sand. The water entering with the feed is 3 to 4 tons per ton of ore, while that required for washing is an additional 2 or 3 tons.

Reverting to the disposition of the riffles, the arrival of the concentrate at the corner renders its separation from the middling a matter requiring considerable attention. Normally such a middling product is taken at the forward end of the bottom edge, the concentrate being taken at, or just round, the corner. Such a division will suffice with a simple ore and a low percentage of mineral, but where greater latitude in drawing the line between concentrate and middling is necessary, the normal riffling is not so satisfactory. Sometimes, then, several of the lower riffles are taken right across the table, to deliver the concentrate at a point on the side, away from the corner. Sometimes, again, all the riffles are continued right across, this disposition presenting the advantage that the clean mineral never gets far down the table, but is delivered high up. The riffle, however, may not be deep right across, or otherwise some gangue, taking advantage of an undeserved protection, will be delivered with the concentrate.

The normal Wilfley is essentially a cleaning table. Provided with deep riffles right across it becomes a 'roughing table,' at a considerably increased capacity. Doing roughing work, with coarse material this increased capacity might be as much as 100 tons or more per day.

Ferraris or Buss Table.—Of similar effect but with different movement, riffling, and support, is the Ferraris table, designed originally on the Continent. The movement is that arising from the alternate pull and thrust of an eccentric attached by an inclined connecting-rod to a table supported on lath-like springs lying over towards the eccentric at an angle of about 75° ; it is in fact the Ferraris movement previously mentioned under screening (Fig. 136). In response to it, the material fed into a fixed feed-box of small dimension hops rapidly across the table. In relation to velocity-acceleration the forward and backward strokes of this movement are not different ; they differ only in that the forward stroke is an upward stroke at the end of which the material is sent flying forward, to be out of contact with the table during the downward and backward stroke. With so vigorous a progression good opportunity must be provided for the wash-water to remove the gangue, and to that intent the riffles of the Buss table are spaced widely apart, diminishing the protection (Fig. 266). Nor are they straight; from a wider spacing at the rear end they converge somewhat, and then take a slightly downward course forward to the concentrate discharge. Number and amplitude of stroke are similar to the figures given for the Wilfley table; the speed is perhaps a little greater and the amplitude a little less. The actual stroke is adjustable by altering the length of the connecting rod, this being possible by means of a turnbuckle on that rod.

The support upon a number of inclined wooden springs standing up from an iron frame, is one which has since been widely adopted; the attachment of these springs with the deck is an elastic one, that with the frame beneath is rigid. The wear on such a support is little, as also is the power consumed in the movement, since no parts move or rock upon one another; nor is any lubrication necessary. Moreover, these springs may be regularly distributed under the deck, directly supporting every portion and permitting a lighter construction. The iron frame below may be either rigidly fixed to the foundation, or so supported that the transverse slope of the table may be adjusted.

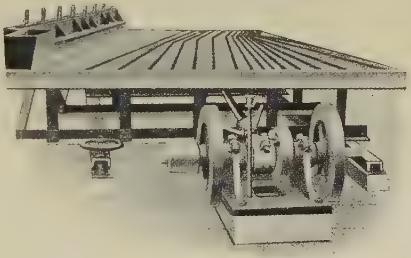


Fig. 266.

Buss Table.—General View (Sandycroft). This illustration shows plainly the rows of springs which support the table, there being probably 20 springs in all. These springs rise from transverse runners which lie on longitudinal channels; it is seen that the tilt of the deck can be altered. The connecting rod to the table is shown, together with two handles of the turnbuckle by which its length may be adjusted; the flywheels on the driving shaft are a noticeable feature. On the deck, the small size of the feed-box indicates that no wide distribution is necessary; the arrangements for washing are noticeably complete; the wide spacing of the riffles likewise favours washing (p. 366).

This table is of trapezoidal shape, 16 ft. long by 7 ft. 6 in. wide at the rear end, and 4 ft. 6 in. at the forward end. Of such dimensions, it has the capacity of 8 to 10 tons of fine sand per day, requiring about 5 tons of clear water per ton of ore. The Ferraris table is similar (Fig. 267).

James Table.—The James table, driven with a toggle mechanism of the same general class as the Wilfley, and supported upon inclined springs in a manner similar to the Buss, differs from them both: firstly, in that the motion is not directly across the stream but diagonally at an angle of about 30° to the bottom edge, the riffles lying at the same angle

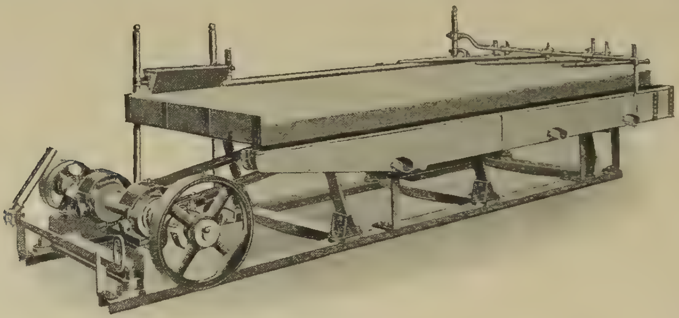


Fig. 267.

Ferraris Table.—General View (Krupp). The number of supporting springs is small, the deck being substantial enough to do without a great number. The two eccentrics make perhaps a better drive; no means of altering the length of the connecting rods is apparent; again there is a flywheel on the driving-shaft. The feed-box is small; it is seen to be fixed to the underframe. The wash-water arrangements are complete, distribution being by pipes. The riffles are straight and short, not reaching to the concentrate end (p. 367).

(Fig. 268); secondly, in that the deck surface is not a single plane, but from top to bottom is constituted of various planes, the first, unriffled for

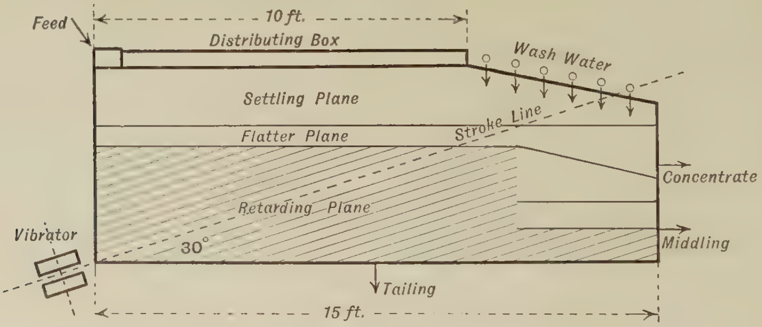


Fig. 268.

James Table.—Diagrammatic Plan of Slime (Fine Sand) Table. The long distributing-box is not fixed to, nor does it move with the deck (p. 368).

mineral settlement, the second unriffled and flatter for collection, and the third a retarding plane covered with riffles, these several planes being repeated on a smaller scale at the concentrate end, to the cleaner separation of the products. The effect of the diagonal disposition of stroke and riffles is that any heavy mineral which may have descended so far as the retarding plane, is thrown back again.

With shape and dimension very similar to the other two tables, the James table makes a faster stroke and one of smaller amplitude. Such a stroke being less effective in distributing the feed, this table is provided with a long feed-box fixed to the frame and commanding more than half the table length. Treating a proper material a very pretty and effective separation is made; the material must, however, be properly prepared, sand must go to a sand table of narrower width and suitable movement, and slime to a slime table.

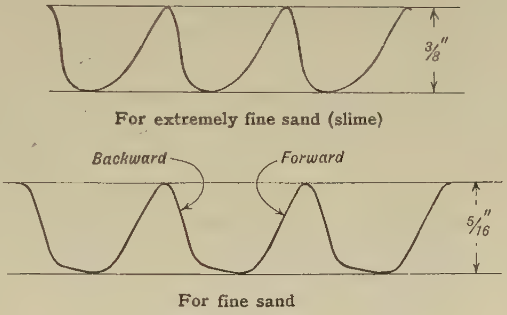


FIG. 269.

James Table.—Graphs of Stroke. The smartness of the backward stroke is pronounced (p. 369).

A graph of the stroke shows the forward movement to be slower than the backward, these two curves being asymmetrical with respect to the vertical ordinate through the forward point (Fig. 269). With slime, this forward point is more rounded, and with sand more pointed. The stroke is generally less than half an inch and the speed greater than 300 per minute, justifying the name of vibrator for the head motion.

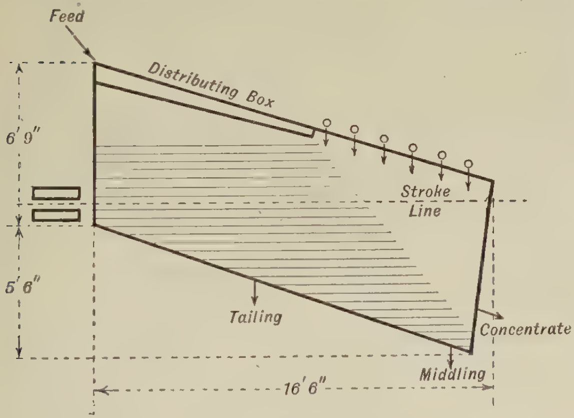


FIG. 270.

Overstrom (Deister) Table.—Diagrammatic Plan. The distributing box is fixed to and moves with the table. It will be noticed that the concentrate discharge is kept wet by its oblique bearing to the stream of wash water (p. 369).

Overstrom Table.—A diagonal relation of the riffles to the bottom edge is also a feature of the Overstrom table, but the stroke and riffles on this table are not at the same time oblique to the stream, as with the James table, but directly across it (Fig. 270);

the diagonal relation with the bottom edge is obtained by laying this edge diagonally to the stream. This table, in fact, is designedly extended along the diagonal course marked by the riffle ends; of great relative length along this diagonal it has many more riffles than usual, and generally about one hundred. Granted each riffle bed to be an effective concentrator in itself, this table should give a very clean product. Its stroke is made by toggle mechanism, amplitude and rate being much the same as with the machines already described.

The Deister Table.—Another design for the concentration of very fine material employs stratification in an extended pool or basin, rather than behind riffles. This design was adopted in the Deister Slimer, which has a square shape to contain the necessary pool (Fig. 271). The pulp arriving on this table at the rear corner is distributed along the top and around the forward corner, flowing thence diagonally down a smooth slope on which settlement takes place, into a pool about half an inch deep, formed by a riffled dam parallel with the centre line of the machine. Along the riffles of this dam the concentrate settled in the pool creeps towards the concentrate discharge. The material which, in the meantime, has overflowed the dam, suffers a similar concentration in a secondary pool in the lower half of the machine, the concentrate from which moves parallel to that from the first pool. Before actual discharge, the concentrate from both pools passes under a series of water-jets by which it is cleaned, the wash water carrying away a middling product at the bottom corner. The eventual tailing is discharged over the lower dam.

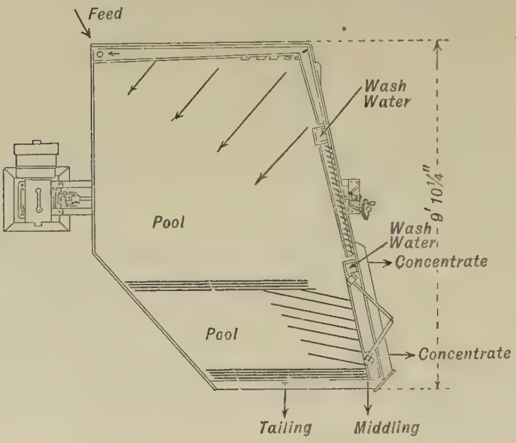


FIG. 271.

Deister Slime (Fine Sand) Table.—Outline Plan of Left-hand Table. The feed-distributing box extending along the upper side and round the forward corner, forms part with the deck, moving with it. The upper half of the water distribution is likewise attached to the deck, while the lower half is carried upon the frame, its actual position being adjustable. Opposite the head motion, on the other side of the table, is seen the handle by which the tilt of the table is adjusted (p. 370).

The head motion of this Deister machine is an interesting piece of mechanism (Fig. 272). After a forward stroke the table is brought back by the depression of the long arm of a bell lever, to the short arm of which it is connected. This depression is caused by the revolution of an eccentric. On the passing of the eccentric the table is taken forward again by the extension of a spring compressed when the table came back, the rate of this forward movement being governed by the speed at which the passing eccentric allows the lever to rise. This engagement of eccentric and pivoted lever results in a difference between the forward and backward strokes very similar to that produced by the Wilfley mechanism.

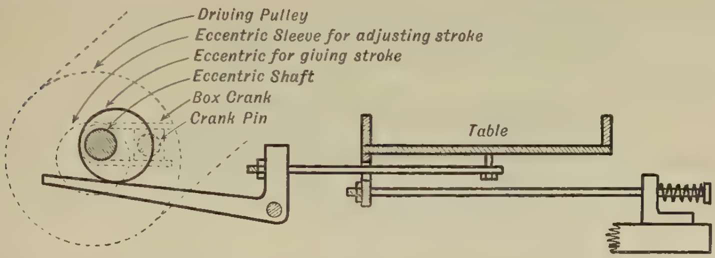

Driving Pulley
Eccentric Sleeve for adjusting stroke
Eccentric for giving stroke
Eccentric Shaft
Box Crank
Crank Pin
Table

FIG. 272.

Deister Table.—Diagram of Mechanism. The length of stroke may be adjusted by raising or lowering the end of the tie-rod held by the short arm of the bell lever. The character of the stroke may be altered by shifting the position of the eccentric sleeve around the shaft; the position shown is the normal setting. The crank-pin is held by brass bushings, which slide within the box-crank to meet the varying distance between crank-pin and shaft, as the driving pulley revolves (p. 371).

In addition, a further differentiation between the two strokes is obtained by driving the eccentric shaft at a varying angular speed, the backward stroke beginning at an increased velocity. To accomplish this, the driving pulley is not keyed to the eccentric shaft, but runs loosely upon an eccentric sleeve interposed between pulley and shaft, the movement of the pulley being conveyed to the shaft by a crank-pin engaging a box-crank keyed to the shaft. Since this pin is on the driving pulley it moves eccentrically with respect to the shaft, being sometimes nearer and sometimes farther away from the axis of that shaft. When nearer, it gives that shaft greater angular velocity than that at which it itself is moving;

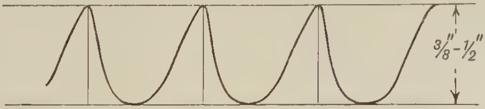


FIG. 273.

Deister Slime Table.—Graph of Stroke (p. 372).

when farther, less angular velocity. The exact position of the eccentric sleeve, while capable of adjustment, is normally so set that the greatest angular movement is at the beginning of the backward stroke. The eccentric itself and the box-crank are at the same angular position upon the shaft.

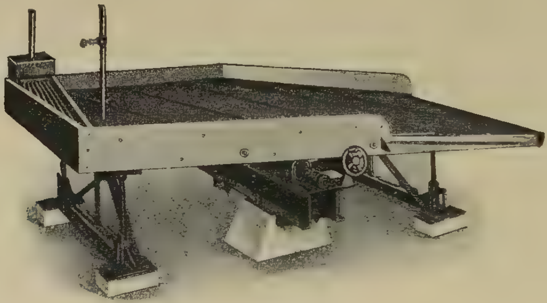


Fig. 274.

Deister Slime Table.—General View. The right-hand table illustrated has no pool. The manner of support upon rocking rods is seen. These rods are borne upon wedge blocks capable of being moved to and fro, by turning the handle shown. As the back wedges are inserted the front wedges are withdrawn, and vice versa ; in this way the table may be tilted to a desired inclination. Near the tilting handle the spring which effects the forward stroke is seen (p. 372).

The normal graph of the movement shows a pointed reversal forward and a rounded reversal at the rear ; in addition, it shows the line of the backward stroke to be the steeper of the two, indicating the greater speed at which that stroke is made (Fig. 273).

The Deister table with a pool does not appear of recent years to have maintained the favour it first secured (Fig. 274). More recently a table of the same name has been introduced which, having riffling similar to the Wilfley, throws its concentrate up on to a higher plane known as the ‘Plateau,’ across which, and under wash water, it makes its way to the concentrate discharge (Fig. 275). In climbing to this higher plane the denser particle has the advantage. Arrived at the top, the particles, which had become banked upon the

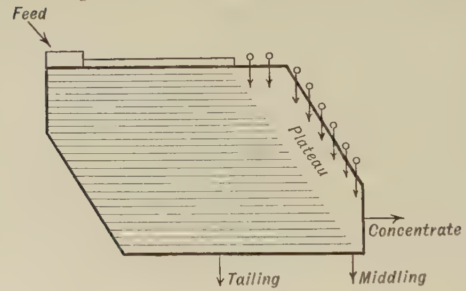


Fig. 275

Deister Plateau Table.—Diagram. Some Deister tables of this type have a more rectangular shape (p. 372).

incline, shoot forward again, separately and individually, each to run the gauntlet of the wash water, whereupon any gangue particles, now caught without the support of heavier neighbours, are washed away. This particular design has received the endorsement of adoption.

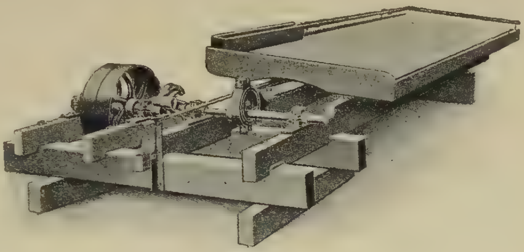


FIG. 276.

Record Table.—General View (p. 373).

Record Table.—The Record table differs from those previously described, by its greater length and width, these being respectively 18 ft. 6 in. and 7 ft. 6 in.; by the great amplitude of its stroke, this being about 3½ in.; by its support upon horizontal slides, in the place of springs or rockers; and by the mechanism of its head motion. Otherwise, its riffling has no special features, while feed and discharge take place around its rectangular outline much as with the Wilfley table (Fig. 276). The differential motion is given by an eccentric on a shaft driven at a variable speed around the circle, by another shaft parallelly disposed but about 3 in. out of line. Cranks on these two shafts, one on each, are connected by a drag link, this connection converting the uniform angular velocity of the driving shaft into variable angular velocity round the eccentric shaft; the eccentric itself would give a harmonic motion, a purely oscillating motion (Fig. 277). Though the rate of these movements is not above 100 per minute, the amplitude is so great and the table so large and massive, that the power consumed is larger than usual, approaching 2 h.p. instead of being less than 1 h.p. The mechanism is, however, cheap and strong, and the table simple, so that it is neither costly to purchase nor

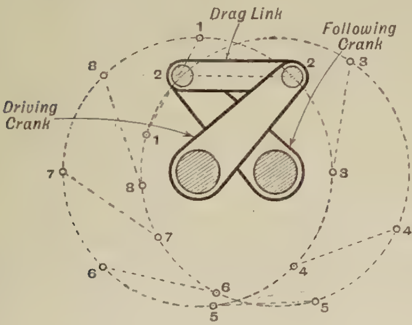


FIG. 277.

Record Table.—Diagram of Mechanism. The irregular intervals between the figures 1 to 8 around the circle described by the following crank are those which result from uniform movement of the driving crank. This irregular movement round the circle is conveyed to the table by an eccentric on the following shaft. The distance between the two shaft-centres is about 3½ in.; the link is a little longer than this distance (p. 373).

to repair. With sandy material it has a capacity commensurate with its greater area, the large throw effecting a quick progression. Contrary to what might be expected, this large throw is said not to produce undue disturbance. The table, however, does not save fine material, and its place would appear to be as a roughing table, to separate an enriched product from a tailing not worthy of further treatment.

Butchart Table.—Appearing more recently, the Butchart table, the essential feature of which are its curved riffles, has been well received. On a deck of normal rectangular shape, the riffles, starting from the feed end, run parallel to the discharge, that is to say, directly across the stream, till they reach the diagonal zone made familiar by the Wilfley, when instead of continuing an unbroken straight course, they curve upward across this zone before again making straight for the discharge (Fig. 278). In the curve upwards, the material which hitherto has rested quietly under the protection of a riffle, receives the force of wash water deflected down the riffle, against which water it has to climb. Only the denser particles survive this ordeal. The material which resumes its course to the discharge is clean concentrate, the gangue having been washed down with the stream. The surface of this table is accordingly divided into three definite areas, the stratifying area directly under the feed, the cleaning area of the curved riffles, and the discharging area beyond. Each riffle, by its traverse of these areas, is a complete and separate concentrator in itself. With cleansing so favoured, this table has a large capacity. Its advent completely established what had hitherto been regarded as abnormal practice, namely, the use of the concentrating table as a roughing machine, treating quantities up to and sometimes substantially exceeding 100 tons a day. To-day it may even be that tables while treating such large quantities can, at the same time, produce clean concentrate as well as clean tailing.

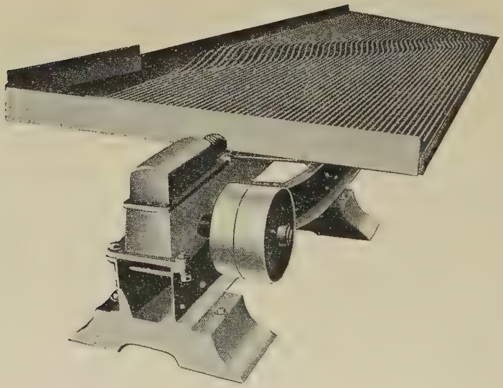


Fig. 278.

Butchart Table.—General View showing Curved Riffles (p. 374).

In addition to its characteristic curved riffle the Butchart table, in common with most recent tables, includes many widely-adopted improvements (Fig. 279). The head motion, for instance, is an enclosed and

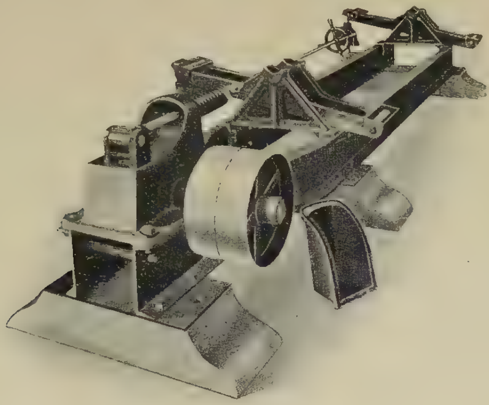
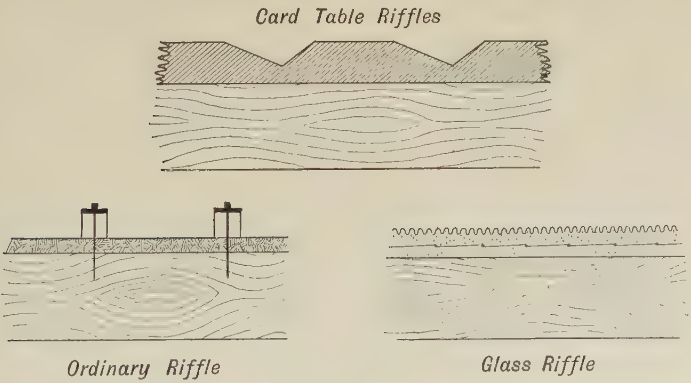


Fig. 279.

Butchart Table.—General View of Mechanism and Base. The mechanism is of the toggle type, the forward motion being made by a spring the tension of which can be adjusted. The tilting frame and the slipper bearings are well shown (p. 375).

self-oiling piece of mechanism; the deck is borne in slipper bearings on a tilting frame, etc.

Card Table.—The Card table differs from those previously described in



Card Table Riffles

Ordinary Riffle

Glass Riffle

Fig. 280.

Riffles.—Card-table Riffle; Ordinary Riffle; and Glass Riffle (p. 376).

that, instead of having riffles nailed to its otherwise plane surface, the opportunities for stratification are provided by grooves sunk below that surface (Fig. 280). These grooves, moreover, have not the square section associated with ordinary riffles, but a flat angular section with a gentle slope on the upstream side and a somewhat steeper slope downstream, this design being to avoid the eddying which square riffles unavoidably set up. Like normal riffles, while deep at the feed end of the table these grooves vanish at the concentrate end. Going forward in the gradually-contracting groove, the stratified material is gradually raised into the cross-flowing current, and thereby cleaned. The head motion of this machine is of the toggle type arranged to give a quick backward movement. The deck plan is, roughly, rectangular, though the length at the top is somewhat longer than at the bottom. Concentrate is discharged over the whole of the forward edge, and accordingly it is easy, upon inspection of the products, to draw any desired line between concentrate and middling. For this reason the Card table early found favour.

Tables, General.—In construction, tables must have an impervious top, or otherwise water would soak through to rot the woodwork and soak the floor. Linoleum serves very well ; in addition to being impervious it is durable and light ; it also exercises the necessary hold upon the mineral ; and is a surface upon which riffles may readily be tacked. Sand concrete has sometimes been used, but then, wooden nailing-strips are necessary as seats for the riffles ; concrete tops, moreover, are heavy. Wired-glass tops with die-impressed riffles have also been suggested (Fig. 280). But nothing seems more satisfactory than linoleum. The riffles are usually of wood, though when shallow they may be of metal strips. The wood used is generally a soft pine, but where the feed is coarse, oak riffles last longer and are better. Under ordinary circumstances a table top will last for two or three years.

In the mill, the rectangular shape of the table permits a number to be conveniently disposed in rows occupying a minimum floor-space and presenting conveniences for driving from line shafts. A floor devoted to tables makes an effective display. Though costing rather much to purchase, tables incur relatively little expense in maintenance and repair—these items together being something of the order of 3d. to 6d. per table per day—and consume little power.

In any scheme of water concentration they follow the jigs if such be present. Recent improvements in tables, and particularly the use of tables for roughing purposes, have indeed enabled the table to encroach upon the work formerly done by fine jigs (Figs. 231, 255). Tables, nowadays, can do more work per unit of floor area than fine jigs, than which also they use considerably less water and require less skilled attendance. Tables now will treat an unclassified feed as coarse as 4 mesh, delivering a clean concentrate when the ore is simple, but more often, from such large material, an enriched product to be retreated. Treating finer material and material which has been classified, they normally make a clean concentrate, a clean tailing, and a middling, the amount of this last being an index to the excellence or otherwise of the work done. The middling product is a safeguard against too great a loss in the tailing when the conditions are set to make a clean concentrate; the smaller the bulk of this middling, the better the work done, since the respective bulk of the finished products is greater, and less remains to be retreated. In addition to the three products mentioned, a further product may sometimes be separated from an unclassified feed, namely, the slime which runs directly down from the feed and leaves the table towards the rear end; this use of the table as a de-slimer is, however, not common.

Tables are undoubtedly the most important appliance in water concentration. They are used everywhere, and upon every kind of ore. With base-metal ores they are often responsible for the largest portion of the concentrate recovered. With precious-metal ores they sometimes serve, as at Cobalt, Ontario, to take out very rich material too coarse for satisfactory recovery by cyanidation or flotation, and sometimes to withdraw such base sulphides as would interfere with the proper working of cyanidation.

It does not appear that flotation will diminish the importance of table concentration; tables are, indeed, often used to recover coarse mineral which has escaped flotation (Fig. 329), or sometimes to separate two or more minerals which have been obtained together in a flotation concentrate (Fig. 315). It is more likely that whether pressed by flotation or not, table concentration will increase at the expense of jigging. It must be remarked, however, that, at Broken Hill, New South Wales, and at Anaconda, Montana, jigs have been retained ahead of flotation while tables in that position have been discarded.

Neither round machines nor belt machines can be said to be represented among jerking appliances. It is true that, as has already been mentioned, a round table, the Bartsch, is given a similar though gentle knock, but this knock is only to stratify the material and not with any idea of imparting progression to the concentrate, progression being effected by the revolution of the feeding and collecting services. Of belt machines, also, one or two, the Luhrig and the Monell, for instance, include a small differential movement in their design, but this movement again is for stratification and not for progression, and these particular appliances have been described under oscillating plants, being in fact ordinarily known as vanners.

Representative Horizontal Water-concentrators

	Streaming.				Oscillating.		Jerking.	
	Surface.				Surface.		Surface.	
	Plane.	Rough.	Mineral.	Riffled.	Plane.	Riffled.	Plane.	Riffled.
Rectangular	Frame Inclined table	Canvas table Blanket strake	Tye Box buddle	Sluice box Sluice table	Shaking table ..	Shaking table ..	Rittinger table ..	Wilfley table ..
Round	Revolving table	..	Round buddle	..	Sperry table	..	Bartsch table	Pinder table
Belt	..	Canvas belt	..	..	Frue vanner	Frue vanner, with corrugated belt	Luhrig vanner	..

THE WATER-CONCENTRATING SYSTEM OR FLOW-SHEET

The scheme of treatment represented by the arrangement, number, and types of the machines assembled to dress ore, constitutes the Concentrating System, a system displayed graphically by the Flow-sheet (p. 645).

Within the overruling purpose of dressing, namely to prepare a product acceptable to the metallurgist, the point to be held in view in devising a system is the maximum monetary advantage to be obtained ; there must be the highest possible capacity and recovery from the minimum equipment. Presuming the deposit to be in full exploitation, the equipment, both in extent and type, will depend chiefly upon the character and value of the ore, but partly upon the manner in which the machines are arranged.

If the ore were coarse grained, then after a relatively coarse crushing, it would be divided into classes for separate coarse, fine, and slime concentration, respectively, this scheme, whole, in part, or further extended, representing “ class concentration ” (Figs. 208, 222, 282). The tailing from the coarse class, and perhaps that from the fine, would then be crushed further, the crushed materials being submitted to fine concentration and slime concentration, respectively, this procedure, possibly continued until all were treated by slime concentration, constituting “ stage concentration ” (Fig. 285). Accordingly, class concentration is applied to the whole material, and stage concentration to the originally coarse portion. Moreover, since each class may be further divided, the number of classes treated separately may be considerable (p. 313), whereas in practice the number of stages is limited to the three mentioned, namely, the coarse, fine, and slime stages.

Such a combination of class and stage concentration avoids the wasteful and useless comminution of coarse mineral ; it also avoids the contamination of the whole concentrate which would result if all the ore were ground fine before concentration began; while finally it permits worthless material to be discarded at a coarse stage, with consequent saving in crushing.

Where the ore was medium grained, the principle of recovering the mineral as coarse as possible, though obviously not of the same moment, would still be applied. Employing only fine and slime concentration, an adequate and sometimes high recovery is nevertheless possible from such ore. Much the same may be said in respect to the recovery from the intermediate classes and stage.

Finally, when the mineral was finely distributed through the ore and comminution was of necessity complete, the capacity of the appropriate water-concentrating machines would be seriously diminished, the concentrate obtained would be much less clean, and the recovery low. The end classes and stage of concentration would be even less satisfactory.

To fix the ideas: if the recovery from coarse concentration were 95 per cent, as it very well might be, that from fine concentration might be expected to be 85 per cent, while that from slime concentration would hardly reach 40 per cent. Similarly, where the recovery from coarse-grained ore was over 90 per cent, that from medium-grained ore would probably be less than 80 per cent, and that from fine-grained ore less than 60 per cent. These figures expose the inability of water concentration to treat slime.

Between complex ores containing more than one valuable mineral, and simple ores, for equal recovery and cleanliness the former would require the more diversified and extended treatment. Between rich and poor ores the former would warrant more extended treatment, permitting better recovery and cleanliness.

The arrangement of the machines within the several classes may likewise vary. Normally, the machines in one concentration class are arranged "in parallel," a portion of the same stream being distributed to each, all producing similar products. The advantage of this arrangement is that the amount of material passing to one machine is relatively small, and the machine, in consequence, works under conditions favourable to the making of clean concentrate and tailing. Often, however, machines are placed "in series," clean concentrate and worthless tailing only resulting after two treatments. Such a treatment in series may mean either the retreatment of the tailing, or the retreatment of a rough concentrate, the former being known as "retreatment concentration" (Fig. 256) and the latter as "roughing concentration" (Figs. 226, 255).

Retreatment concentration, though not often clearly defined, is not uncommon. With jigs, for instance, the material passes from cell to cell, each cell being in reality an independent machine making its own products, the material passing forward for retreatment in the next cell. With sand

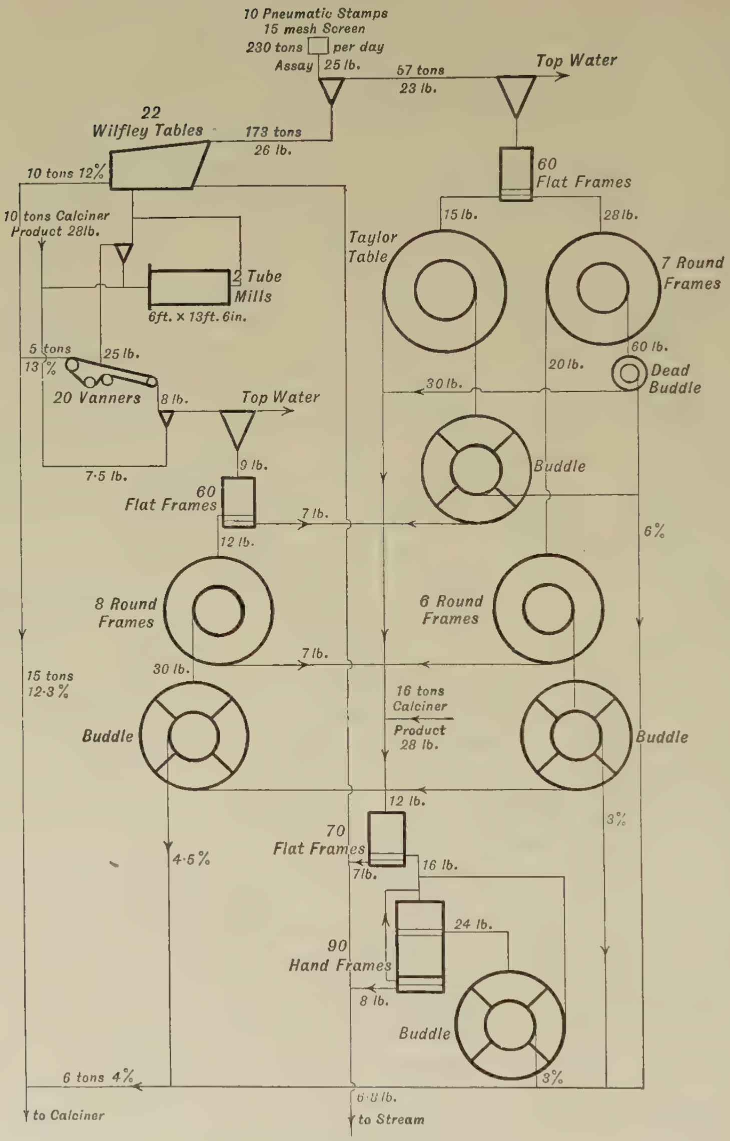


FIG. 281.

Water Concentration, East Pool, Cornwall, 1918.—The flat frames are ordinary ragging frames. The hand-frames are frames worked by hand instead of by an automatic balance-box, two being in series, the lower treating the tailing from the upper. The Taylor table consists of a number of flat frames arranged to work automatically round a circular track. A dead-buddle is one with no moving sweeps or parts, the pulp, generally an enriched product, entering through a downright pipe at the centre and making its own radiating bed upon a flat cement floor about a foot below the surface. After being calcined, further treatment on tables, in buddles and kieves, eventually brings the concentrate to a marketable condition. This further treatment takes place in the 'tin-yard.' The ore is hard, fine-grained and complex, containing cassiterite, wolframite, and mispickel as valuable constituents (pp. 343, 380, 647). (Truscott, *Trans. I.M.M.* Vol. XXVIII., 1919, p. 73.)

tables, retreatment in simple series is not often seen, that is to say, though primary tables are often followed by secondary tables treating a middling product from the primaries, it rarely occurs that the whole tailing passes in succession over a second table; tables are usually too expensive to apply to impoverished material. Finally, with slime tables, it is not unusual to see the tailing from one machine retreated on a second; it is, indeed, only the great floor-space occupied by slime machines which limits such retreatment (Fig. 281).

In roughing concentration, a primary machine, by discarding clean or worthless tailing, prepares an enriched product for treatment by a cleaning machine. Concentrating machines have, largely for convenience, been distinguished from classifying machines, the latter being regarded as purely preparatory; when, however, a concentrator is used as a roughing machine, the work it does is likewise largely preparatory, generally, indeed, to the extent that previous classification is eliminated (Fig. 225). A roughing treatment is in effect an extension of the idea of making a middling product.

Normally, the concentrator is regarded as a machine for making a finished concentrate, and its success is judged by the weight of finished concentrate it produces. So regarded, the taking of a middling product is not an end in itself but a guard against loss in the tailing; such a middling may not be too bulky or it becomes a burden to the system. When this middling consists largely of "chats," that is, of particles of mineral and gangue intergrown, as it mostly does from jigs, it is comminuted further and introduced again at a point lower down the system. When, on the other hand, the particles of mineral and gangue are free, as they often are from tables, it is common practice to send the middling over similar machines suitably adjusted, one such middling machine serving a number of primary machines. To treat middling by returning it to the machine which made it, is bad practice, since this could only result in an accumulation detrimental to the proper functioning of the machine; eventually, of course, such middling would find its way partly to the tailing, partly to the concentrate, to neither of which might it belong.

The partly finished product of a roughing machine is, however, in different circumstance to the middling produced by a finishing machine, being so abundant in amount that the system is specially arranged to accommodate it.

Roughing may be applied throughout the system. With jigs it is sometimes, as at Joplin, the concentrating system itself; with tables, the roughing table, by reason of its large capacity, has notoriously replaced jigs for the treatment of such sand as contains little clean concentrate to recover; with slime tables, roughing permits the discard of a substantial amount of worthless tailing, as do the ‘ragging’ frames so noticeable in Cornwall.

In general, roughing, in addition to simplifying the preparatory operations, offers the advantage that by treating coarse and fine, sand and slime, together, the latter receives treatment and is substantially impoverished before it reaches its own particular tables. This may happen to such an extent that the greater portion of the recovered concentrate of slime size is obtained from the sand tables. With rich material roughing has no place, largely because the desired cleanliness of concentrate is obtained in a single treatment, but also because the value of the ore permits adequate preparation by sizing and classification. Its benefits are chiefly felt with low-grade material.

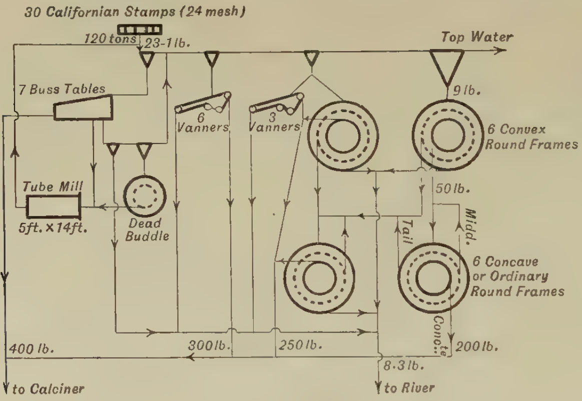


FIG. 282.

Water Concentration, Giew, Cornwall, 1918.—Though treating a simple ore, the pulp from the stamps is divided by classifiers into four different products, these being separately treated on tables, vanners, and frames, respectively. In this scheme also, enrichment of the slime concentrate is made more rapid by taking a middling product from the round frames (pp. 282, 378, 646). (Truscott, *Trans. I.M.M.* Vol. XXVIII.)

A roughing treatment connotes gradual enrichment; normal treatment and retreatment connote rapid enrichment. Where, as with most concentrating machines, separation does not require the enriched product to be accommodated on the surface, gradual enrichment, implying the careful discard of a completely impoverished tailing at successive treatments, is attractive. But where the enriched product must be held upon the surface of the machine, as, for instance, on slime tables, the area of surface required will be greater with gradual than with rapid enrichment, the bulk of the enriched product being greater. In the water concentration of slime, accordingly, a relatively rapid enrichment appears to be demanded by considerations of plant cost and floor-space (Fig. 282).

Finally, where, because of the small amount of mineral in the ore, the weight of the enriched product remaining for treatment is too small to maintain a continuously flowing stream, the system changes to one of treatment in charges. In Cornwall, for instance, the ordinary mill produces a concentrate, which, were copper or lead the valuable metal, might well be sufficiently clean, but, the valuable metal being tin, an almost perfect cleanliness is necessary, and the mill concentrate is sent to the calciner to be roasted, whence it passes to be carefully dressed in charges, in a portion of the plant known as the 'tin-yard.' In that yard it is treated on tables, in buddles, and finally in kieves.

WATER CONCENTRATION, RÉSUMÉ

Successful water concentration requires ponderable size of mineral grain and a sufficient difference of density between mineral and gangue. The highest recoveries are made when the mineral is coarse. The cleanest concentrate often results from sandy material, since with such, as a rule, the mineral release is more complete. With still finer material, though mineral release may be still more complete, the concentrate is likely to be less clean owing to the difficulties of separation from gangue; adequate recoveries of clean concentrate are, however, perfectly well possible from material as small as 200 mesh. Below that size both recovery and cleanliness suffer; the recovery from slime material by water concentration is generally only 30 to 40 per cent.

In respect to difference in density, an adequate separation requires that the specific gravity of the mineral to be recovered should be at least 1·5 higher than that of the gangue, or if there be two minerals present, at least that much higher in density than the lighter mineral.

Recovery and cleanliness are to some extent interdependent, greater cleanliness being generally obtained at some expense to recovery. Cleanliness, of course, is the necessity imposed by the subsequent metallurgical operations, which suffer if it be not achieved. The proper degree of cleanliness reached is therefore determined by the balance between the extra cost and loss incurred in its achievement, and the benefits obtained. It is usual to express the extent to which concentration has been pursued towards cleanliness, by the ratio between the weight of the ore treated and that of the concentrate obtained, that is, by the 'Ratio of Concentration.' This ratio, in turn, depends upon that between the mineral content of the dressed concentrate and that of the original ore, that is, upon the 'Ratio of Enrichment.' These two ratios would be equal were there no loss, since the mineral content of the concentrate would then increase exactly as its weight decreased; as it is, the concentration-ratio is proportionately the higher of the two, on which subject more anon (p. 637).

In the dressing of tin ores assaying, say, 1 per cent of tin, to a concentrate assaying, say, 60 per cent, the recovery is only about 65 per cent, in spite of the high density of cassiterite and the granularity of its particles; the corresponding enrichment-ratio would be 60, and the concentration-ratio close upon 100, this latter figure expressing the fact that out of 100 tons of ore only one ton of concentrate is produced. With rich coarse-grained lead ores, on the other hand, by reason of their considerably-higher original-content, the recovery is much higher, a normal figure being over 90 per cent, and the concentration-ratio much lower, being about 5; a still lower concentration-ratio obtains with iron ores because of the still higher content of these ores. Between such extreme figures come most of the other results. With coarse-grained copper ores recovery will be 80 per cent or so; with fine-grained copper ores, from 60 to 70 per cent, the lower percentage accompanying a higher concentration-ratio, say about 20, the higher percentage going with a lower ratio of about 10. With simple zinc ores, these being generally coarse-grained though relatively low-grade, the recovery by water is about 65 per cent.

Such results as the above are obtained at an operating cost which varies from about a shilling per ton with the simplest ore, to ten or twelve shillings per ton with complex ores, these figures including crushing and classification. All other things—the character of the ore, the concentrating system, etc.—being equal, the precise figure varies with the capacity of the equipment, with large equipments it is substantially less than with small equipments.

The investment cost of water concentration is determined by the prime cost of the installation. Prime cost will vary from £10 per ton treated per day with a simple system, to as much as £150 per ton per day with a system extensively developed. The extent to which the system is developed and recovery is pursued, depends upon the balance between the extra cost incurred and the extra recovery obtained. The investment cost per ton is lowered by using an equipment at full capacity.

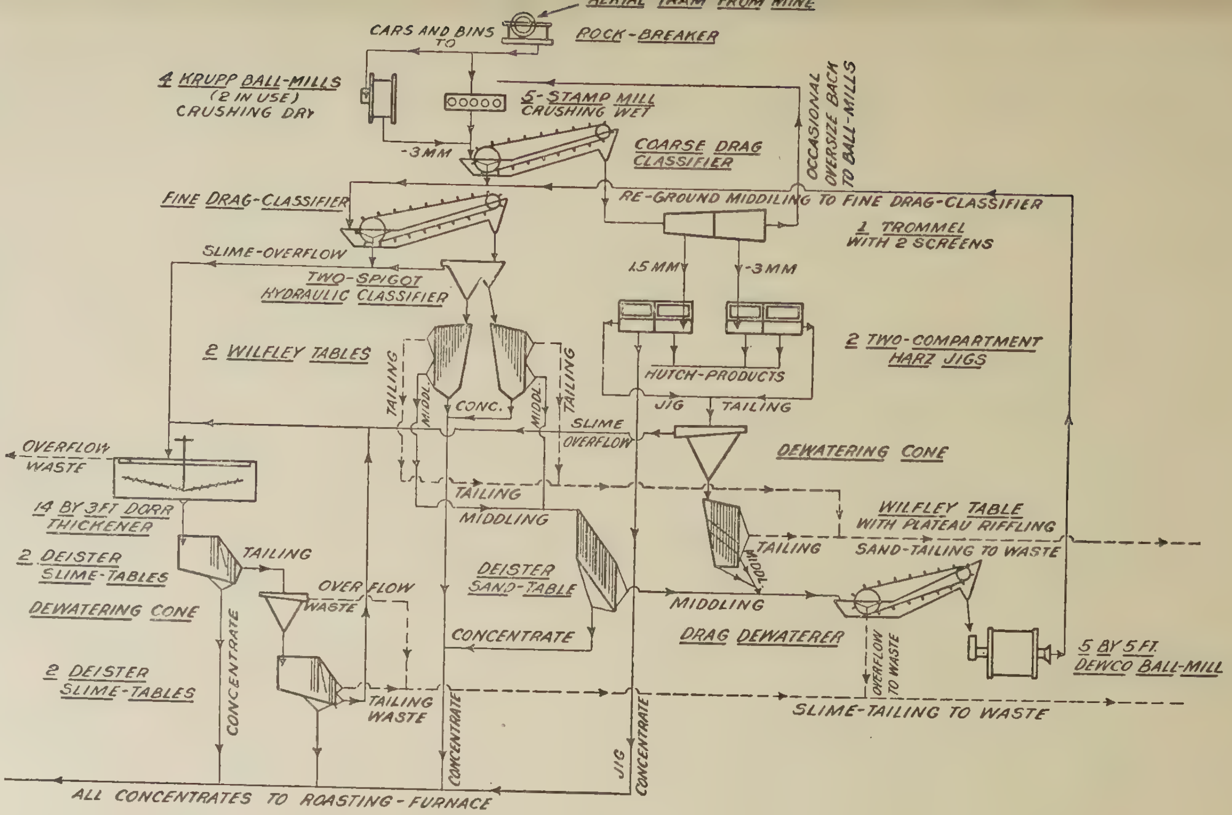


Fig. 283.

Water Concentration at the Avicaya Tin Mine, Bolivia.—Flow-sheet. Cassiterite accompanied by iron oxides, pyrite, and arsenopyrite, occurs in small patches and stringers in an acid eruptive. The feed assays about 5 per cent of tin, and the tailing about 1 per cent. The crude mill-concentrate assays 50—55 per cent, and the roasted concentrate, after washing and magnetic separation, 65 per cent (p. 388). (Söhnlein, *M. & S.P.*, September 8, 1917, p. 343.)

But capacity and recovery are somewhat interdependent; if capacity is pursued, recovery suffers. The endeavour to increase capacity must accordingly be stopped when gain in capacity is more than offset by loss in recovery.

From the above considerations, it is evident that in length of treatment and retreatment the system is dependent upon the original value of the ore. An extra recovery of 5 per cent from a rich ore might well cover the investment and operating costs of another row of machines, for which that additional percentage recovery from a poor ore would not suffice.

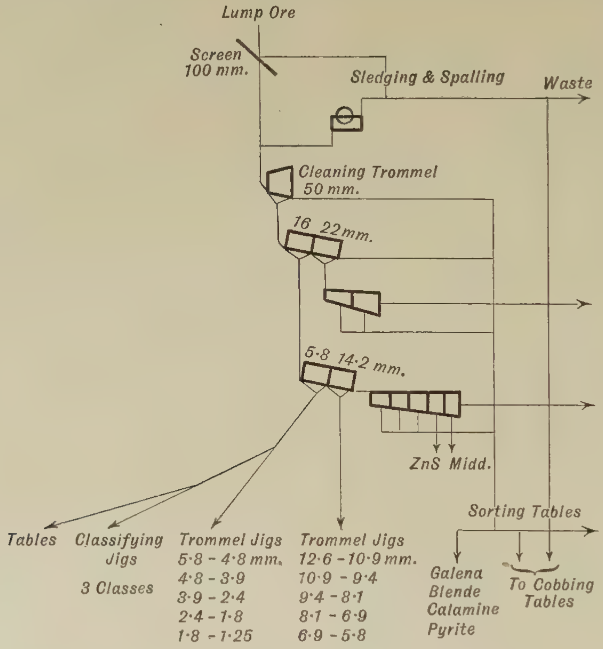


FIG. 284.

Water Concentration, Moresnet, Rhineland, 1916.—The ore treated is a plombiferous blende occurring with pyrite, and associated with dolomite, limestone, and slate. The average ore contains about 5 per cent of galena, 30 per cent of blende, and 20 per cent of pyrite. About 20 per cent of the total ore is removed by sorting, partly as waste and partly as finished products. The recovery of the zinc is about 85 per cent. (pp. 23, 24, 387).

In the matter of capacity, most water-concentrating machines have this favourable point, that, for intervals at least, they are capable of taking an overload, this capability lying in the fact that the bed of material which they retain as essential to their proper functioning, may be augmented. In this bed fluctuations in the feed are absorbed. Fluctuations may well arise, for instance, from the arrival of coarse and hard ore at one time and fine and soft ore at another; from the occasional blinding of screens in the sizing equipment, or of sieves in the jigs. Fluctuations may also arise by variations in the mineral content, even though these may have been largely eliminated by previous sorting (Fig. 284).

The above-mentioned pre-requisites for successful water treatment, namely, ponderable size and sufficiently high density, are largely present in the ores requiring dressing. Quite exceptionally, it is true, they are not sufficient; molybdenite, though it be heavy and occurs large enough, breaks into such pronounced flakes that its concentration by water is impossible; the tellurides and sulph-antimonides of the precious metals,

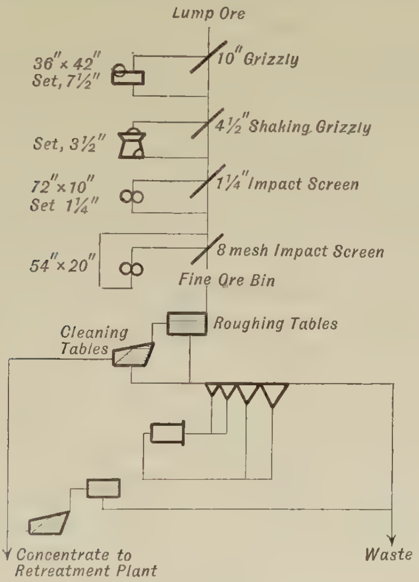


FIG. 285.

Water Concentration, Alaska Gold Mines, 1918.—Flow-sheet. The ore consists of quartz stringers in slate and gabbro, this quartz containing free gold, galena, and auriferous pyrite. During the year 1917 the tonnage crushed was 2,240,000 and the milling cost about 1s. 2d. per ton milled, the yield obtained being about 5s. 0d. per ton. At the Alaska-Treadwell mine working similar ore, stamps were used and not rolls, and amalgamation was practised ahead of the concentration, this latter being largely conducted on vanners; on the other hand, at the Alaska Juneau, with less success, ball-mills were used for the primary reduction, amalgamation was not practised, and about one-third of the ore sent to the mill was sorted as waste. A feature of this milling plant is the retreatment to which the concentrate is submitted, and from which a concentrate of coarse gold, a second concentrate of clean galena, and a third of auriferous pyrite are obtained (pp. 378, 388, 647).

though they be both ponderable in their grain and dense in their substance, are so friable that no adequate recovery can be made by water ; with the oxidized ores of copper, zinc, and lead the position is much the same ; while stibnite occurs so sporadically and is so soft, that water separation is inapplicable.

Yet water concentration has an immensely large field. It is, and promises to remain, the prime and only means of beneficiating tin and wolfram ores (Fig. 283). It is, and doubtless will remain, the treatment for the coarser-grained portion of the base-metal sulphide ores, yielding to flotation only for the finer portion, or where a heavy gangue is present (Figs. 326, 330). It is serviceable in the recovery of auriferous pyrite from low-grade gold ores (Fig. 285), and in the recovery of valuable sulphides, or the removal of harmful sulphides, before the cyanidation of precious-metal ores generally. It removes cheaply and satisfactorily the coarse sulphides of an oxidized copper ore preparatory to subsequent leaching of the remainder. And where flotation is the prime process, it provides a cheap means of recovering such coarse sulphides as escape that treatment (Fig. 329).

In conclusion, to-day probably a greater tonnage is treated by water concentration than by any other process of dressing. It may be that much of this tonnage would not be in a position to be treated by water were it not that flotation assisted and made possible the exploitation of the deposit. Nevertheless, it cannot yet be said, that, as with the passage of time hand-picking gave precedence to water concentration, so water concentration is now in the way of ceding pride of place to flotation.

CHAPTER IX

FLOTATION CONCENTRATION; PROCESSES, MACHINES, AND AGENTS

GENERAL

Development.—We are close enough to the original conception and development of this important process of concentration that an ordered recital of events may be attempted. In this recital something is learned of the properties upon which the process is based.

Water concentration has been practised for ages, and its main principles were well understood away back in the time of Agricola. Flotation belongs to the present era of mining, which dates back to the exploitation of the rich auriferous gravels of California and Australia about the middle of the nineteenth century. The first reference to the affinity of oil for mineral in preference to gangue, and the first suggestion to use this preference for the recovery of the mineral, was made by William Haynes, of Holywell, Wales, in 1860; it cannot be said seriously that any such suggestion had been made before. Haynes recommended the addition of an oily mixture to powdered ore in the proportion of one part by weight to about six parts of ore, the ore to be kneaded till the fatty mixture had exerted its preference for the mineral, agglomerating it into a pasty mass from which the unselected gangue could be driven by water. In this recommendation there was no suggestion of flotation; nor was there any continuity in the process, the material was treated in charges. It is interesting, however, that fatty or oily matters, coal tar, and gum, were enumerated as suitable selective agents.

A different idea was that put forward in 1885 by Hezekiah Bradford in the United States, who described a method for the treatment of wet tailing from water concentrators, by passing such tailing in a thin stream down a slope on to the surface of water in a pointed vessel, the sulphides floating and overflowing, the gangue sinking. This continuous flotation process depended upon the non-wetting properties of sulphide minerals.

Then, in direct line to Haynes, came Carrie Everson of Chicago, who in 1885 suggested the mixture of oil and acidulated water with crushed ore to form a stiff paste from which, again, the gangue could be removed by water, leaving the sulphide mineral behind. In this recommendation the particular contribution to the art of dressing was the increased selectivity brought about by the acid. But somewhat later, and in association with others, the suggestion was made to conduct the operation in the presence of sufficient water that the oil, aided by gaseous effervescence and some aeration, floated the mineral to thé water surface, separating it thereby from the gangue, which remained below. This process, though flotative, was not continuous but proceeded in charges.

In 1894 Robson, of Dolgelly in Wales, with Crowder, of London, agitated moist ground ore in a vessel through which a stream of thin oil flowed upwards, this thin oil being made by the addition of a solvent hydrocarbon, turpentine or kerosene, to a thick fatty oil. In this upward progress of the oil the sulphide particles were entrained and carried to the overflow, the gangue subsiding. Though, therefore, the mineral particle rose, flotation by buoyancy was not unaided, the stream assisted; nor was the process continuous.

The first flotation process to be commercially applied was, however, that developed by Elmore of London, in 1898. Elmore was, moreover, the first to apply the selective action of oil to a pulp flowing freely from a wet-crushing machine. This freely-flowing pulp entered a cylindrical mixer revolving slowly round a horizontal axis, into which cylinder and at the same end a thick mineral-oil was introduced (Fig. 286). From the other end the oily mixture discharged into a spitzkasten, wherein the water-level was so maintained that the floating oil overflowed separately and continuously, carrying with it the mineral particles, while the unaffected gangue was discharged below, with the water. The recovery of the mineral from its oily menstrum was then effected in centrifugal machines, the oil being returned to service. In respect to the amount of mineral, the amount of oil present, lifting only about 10 per cent of its weight of mineral, was considerable. But since, after centrifugal separation, this oil was returned for further use, the amount consumed was only that inseparable from the concentrate and that lost in handling, this consumption amounting to about 2 per cent of the weight of the ore treated. In this process, which is commonly referred to as 'The Elmore Bulk-Oil process,' there was complete and natural flotation, no upward stream of oil assisting.

The results obtained by the bulk-oil process at the Glasdir copper mine in Wales were such as to make it quite evident that this new process of mineral concentration had wide possibilities. Hitherto, at that mine, extended water-concentration had made but poor recoveries, owing to the friable nature of the ore-mineral, chalcopyrite containing some gold and silver; but from the same ore oil-flotation at once secured a recovery of about 80 per cent of the copper, silver, and gold present, these recoveries being obtained from a plant having a capacity of about 50 tons per day. In addition to this commercial application, the process also secured equally satisfactory results on a laboratory scale from a variety of ores. Such results focussed the attention upon this new process, and dispelled an attitude of incredulity. Headway did not, however, take the line of direct succession to the ideas which Elmore had so presciently put into operation.

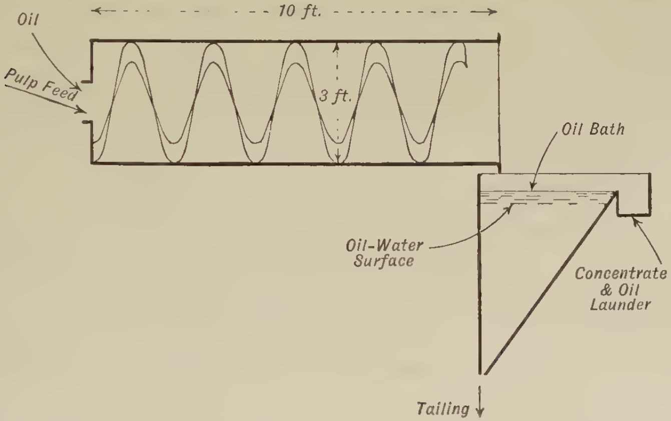


Fig. 286.

Elmore Bulk-Oil Flotation Machine.—Diagram. The material, ground to about 30 mesh, passed with a thick fuel-oil into the end of a slowly-rotating drum lined with helical flights projecting about six inches into the interior. At the other end discharge took place into a pointed box, whence the mineral-laden oil overflowed while the gangue sank to be discharged through a spigot. The time of passage through the drum—determined by the pitch of the helical flights and the speed of revolution, this latter being about five per minute—was generally about 2 minutes. As a rule three such drums worked in series, so that the material received three treatments before final discharge (p. 390).

About this time the zinc middlings resulting from the water concentration of Broken Hill ore, such middlings being untreatable by reason of the amount of heavy gangue present, were awaiting the discovery of some process by which they might be beneficiated. It had long been noticed in the water concentration that at places in the flowing system where the stream became broken and agitated, the surface of the water was often covered with a glistening film of floating sulphide, to sink which had hitherto been the anxiety and endeavour. With the change of attitude towards the possibilities of flotation, came also a change in this endeavour, which hereafter was to get more of the sulphide to float. This was particularly the position about the year 1901, after the possibilities of dry magnetic-separation of the zinc blende had indicated that no satisfactory treatment lay in that direction.

Early in 1902, Potter at Broken Hill evolved a process to treat these zinc middlings in hot acidulated-solutions. From the dump these middlings were introduced into a vessel containing a solution of sulphuric acid maintained at an acidity of $2\frac{1}{2}$ per cent and a temperature not far from boiling-point, and stirred, the zinc blende under this treatment rising to be included in a froth upon the surface, while the gangue fell to the bottom to be continuously discharged. Actually, flotation was effected by the attachment of the blende to bubbles of carbonic-acid gas, generated by the action of the acid upon the calcite in the gangue; it was, accordingly, a gas-bubble flotation and not an oil flotation. This fact was emphasized by Froment later in the year, in a process specifically stated to use gas bubbles, generated by acid from calcite, to increase the buoyancy of particles agglomerated by oil.

Delprat, towards the end of the same year, 1902, applied a similar process at the Broken Hill Proprietary mine, replacing sulphuric acid by salt cake (Fig. 287). By its means, and without the use of oil, the first parcel of clean concentrate resulting from flotation at Broken Hill was obtained, this parcel of 50 tons being marketed in May 1903. Later, by arrangement with Potter, the Delprat plant returned to the use of sulphuric acid, using which it has continued to operate with every satisfaction. It must be remarked, however, that this success was obtained upon a particular material, namely, one containing about 3 per cent of calcite and consisting of fine sand with little or no slime. It depended upon the attachment of the sulphides to gas bubbles, this being an expression of their non-wetting properties. Arrived at the surface with their load of mineral these bubbles formed a thick coherent froth.

Cattermole of London, in 1903, proposed the addition of oil in such amount, namely, about 5 per cent of the weight of sulphide mineral present, that such mineral particles became agglomerated into floccules and sank down through a rising current strong enough to lift the gangue particles to the overflow; treatment included a gentle agitation of the pulp and oil to assemble the mineral particles, separation taking place subsequently in a spitzkasten. The oil used was a heavy mineral-oil, and afterwards oleic acid with soap as an emulsifying agent. It was found, however, that all the sulphide particles were not thus selectively flocculated, but that some, attaching themselves to bubbles, floated to the surface to overflow with the gangue. Later in the same year, and in association with Sulman and Picard, the suggestion was made to use the fatty acid set free by the action of acid upon soap solution, the gas bubbles generated by the action of excess acid upon the calcite being used to float the floccules. The Cattermole process was tried for some time, but in practice was never successful. It depended upon the oiliness of the oil.

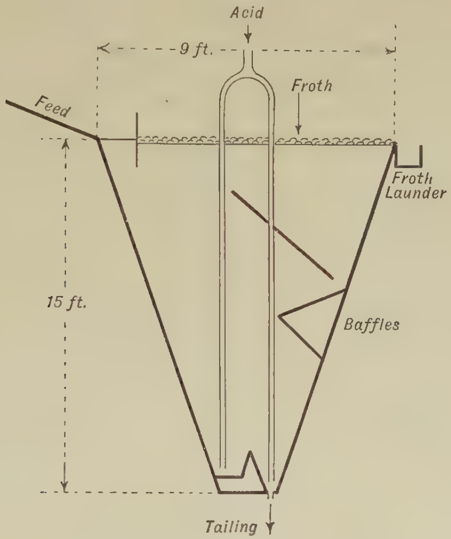


Fig. 287.

Potter-Delprat Flotation Box.—Diagram. Zinc middling from previous water concentration is pushed down a feed sole heated underneath, at the rate of about one-quarter of a ton per minute, into a bath maintained by live steam at a temperature of about 90° C., and at an acidity of about 2 per cent of sulphuric acid. This bath is contained in a cast-iron pyramidal box 9 feet in dimension at the top and 15 feet deep, provided with an overflow launder opposite the feed entry and a spigot discharge for the underflow. This discharge is from one of two pockets arranged in the bottom, the other pocket being blind. Each of these two pockets is separately served with acid by a pipe reaching down from the acid service above. In this way the material receives two treatments, the first in the blind pocket, the second in the discharge pocket. In order that gangue particles entangled with the rising sulphide from this second pocket may not pass directly over into the concentrate launder, baffles are arranged to direct this rising sulphide away from the overflow (p. 392). Representative figures obtained without the use of oil, and without any trace of oil detectable in the concentrate, were :

	Zn.	Pb.	Zn.
	Per cent.	Per cent.	Oz.
Feed	14	3	5
Concentrate . . .	47	6	13
Tailing	3	2	2

De Bavay of Melbourne, in 1904, treated finely-crushed dry ore by delivering it gently on to water, when the sulphides by reason of their non-wetting properties remained on the surface as a thin float, the gangue sinking. This process he further developed by directing an ore pulp over a corrugated conical surface, rolling down which the sulphide particles, as in the turmoil they broke the water surface, rode away, the gangue finding a separate outlet below (Fig. 288). This process, modified by subsidiary aeration and by the addition of a small amount of oil, was long in use after its first introduction, producing the cleanest concentrate at Broken Hill. The scum or float arising in its operation was noticeably different from that produced in the Potter-Delprat process, being but a thin mineral-film and not a thick and coherent froth — a difference which may be taken to distinguish between 'film flotation' and 'froth flotation.' The process was not successful with slime nor with sand containing slime.

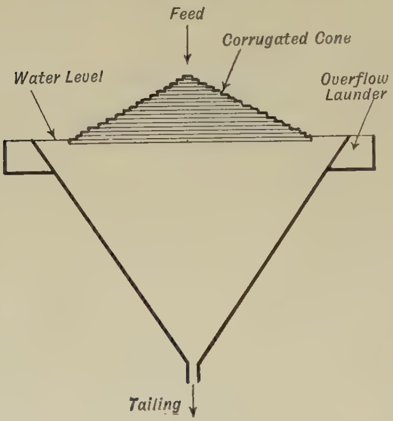


Fig. 288.

De Bavay Flotation Cell. — Diagram. The pulp is fed on top of a corrugated cone supported concentrically within a conical classifier at such a height that its base is just under the water surface. Upon that surface the sulphide forms a film which overflows into a peripheral launder, while the gangue drops below to be discharged successively upon other cones in series (p. 394).

Macquisten of Glasgow, in the same year, 1904, developed a very similar process. Instead, however, of giving the mineral particle opportunity to break the water surface during a troubled fall, he introduced the pulp into one end of a horizontally rotating pipe with spirally corrugated interior, settling wherein the particles became lifted to a water surface maintained below axial level (Fig. 289). Breaking through that surface, the sulphide particles floated while the gangue particles dropped back, the flow of water through the pipe carrying the resultant mineral-film forward, while the spiral corrugations ensured the progression of the gangue. In this process as originally conceived, no oil was used. It was applied commercially in the western United States about 1910 to recover zinc and copper mineral from sandy tailings and middling products.

Then, in 1904, again, Elmore brought out a second process which, while embodying the selective action of oil in a mobile pulp, incorporated the by-this-time familiar idea of floating the oiled sulphides by means of bubbles, these bubbles forming a froth at the surface. The air for the bubbles was obtained by reducing the pressure above the water-surface, the air normally in solution in the water—about 2 per cent of the water volume—then coming out in the form of small bubbles; from this use of reduced pressure the process is known as the Elmore-Vacuum process. The eventual discharge both of the froth and of the tailing was through vertical pipes hanging deep enough to maintain the vacuum and to act as the long legs of a siphon, the short leg being that up which the feed was drawn (Fig. 290).

In 1905, from experiment along lines suggested by the failings of the Cattermole process, Sulman, Picard, and Ballot, together put forward a process employing an amount of oil specified to be less than 1 per cent of the weight of the ore treated and generally about 0·1 per cent, the treatment including a violent agitation. With this small amount of oil, oiliness was no longer a factor; at most the oil could but form the thinnest film upon the sulphide particles, a film roughly of molecular dimensions but sufficient to contaminate the surface. Moreover, in the violent agitation specified, the mechanical emulsification of the oil was so complete that, doubtless, fractional solution supervened and the water became contaminated, permitting in that condition the formation and maintenance of myriads of fine air-bubbles. To these bubbles the sulphide particles

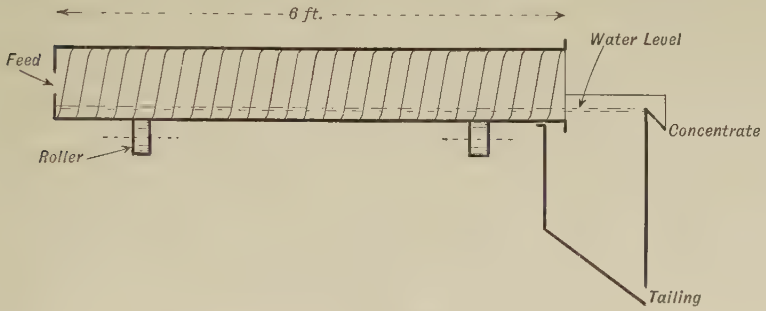


Fig. 289.

Macquisten Tube Concentrator.—Diagram. A cast-iron tube about 10″ in diameter and 6 feet long with helical groove of about 3-inch pitch lining the interior. The material to be treated is crushed to about 30 mesh and the slime separated. The remaining sand is fed at the free end of the tube, which is rotated at about 30 r.p.m. With ordinary ore two tubes in series, the second retreating the tailing from the first, would make a satisfactory recovery and be capable of treating about 5 tons per 24 hours. A 2·5 per cent copper ore would yield a concentrate assaying about 20 per cent. This appliance was used at Golconda, Nevada, to treat a copper tailing containing spinel and garnet; and at the Morning mine, Idaho, to treat a middling containing galena, blende, and siderite (p. 394).

Elmore-Vacuum Plant.—Diagram. The separating vessel is a cone with apex upwards. Through the centre of the base the oiled pulp, having a consistency of about 3 of water to 1 of ore, enters, being drawn up by the vacuum above. Arrived within the cone it is stirred by rotating rakes worked by a worm wheel, these rakes at the same time gradually ploughing the tailing to the periphery, where it is discharged impoverished down pipes extending about 30 feet below. This discharge is under water to keep air out, and so controlled as to leave just sufficient of the feed water to discharge the concentrate. The feed being drawn up about 20 feet, its pipe and the discharge pipe together form a natural syphon at the top of which, even without the vacuum pump, there would be considerably reduced pressure (p. 394).

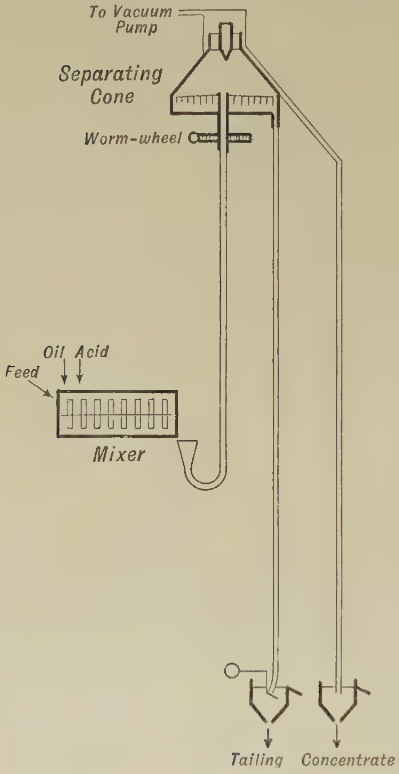


FIG. 290.

During the progress of the material towards the periphery, the oiled sulphide particles escape by attaching themselves to air bubbles engendered by the vacuum. These loaded bubbles rise, the conical shape of the vessel bringing them all to the centre, where they overflow a circular lip into an outer hood from which the concentrate discharge-pipe leads.

The standard machine with a cone 5 feet in diameter, and using about 3 h.p., treats 20 to 40 tons per day, the lower figure applying to ore containing much sulphide. From 3 to 10 lb. of oil is used per ton, and a similar amount of acid; the vacuum employed is about 24 inches. Gas generated by the acid assists substantially.

This process was considerably used at Broken Hill, and until recently was still in use at Sulitelma, Norway. Upon sandy material, even though fine, it made good recoveries and a good enrichment. Representative results are as follows:

	Broken Hill.			Sulitelma.	
	Zinc Middling.			Mill Ore.	Water Tailing.
	Zn.	Pb.	Ag.		
	Per cent.	Per cent.	Oz.	Per cent.	Per cent.
Feed	20	6	8	2·5 Cu.	1 Cu.
Concentrate . . .	43	11	17	17·5 ,,	7 ,,
Tailing	. .	. .	. .	0·25 ,,	0·25 ,,

attached themselves, the loaded bubbles rising to form a dense, coherent, and many-layered froth, armoured by the mineral it contained.

This idea of driving a portion of the oil into solution was further developed by Higgins, who, in 1908, suggested the employment of aromatic hydroxides, such as phenol, as froth-forming agents; and by Sulman, Picard, and Ballot, who, in the same year, put forward the alcohol series as soluble frothing-agents. These suggestions in turn were followed in 1909 by that of Greenway, Sulman, and Higgins, to use amyl acetate, camphor, etc.; by that of Greenway and Lavers to use eucalyptus and other essential oils; and by that of Sulman, Higgins, and Ballot, to use spirits of turpentine, etc.

Of these various processes, the first to secure extended application was the Elmore-Vacuum process, which during the years 1908–1910 treated many hundreds of thousands of tons of sandy zinc-middling at Broken Hill. The zinc concentrate obtained in this treatment, not being sufficiently clean for the market, was given a supplementary water-treatment. To make this effective, the oil remaining in the concentrate was burned off, which being done to the point of incipient roasting, the particles were rendered perfectly mobile in water, with the result that treatment on tables not only yielded a zinc concentrate of the necessary cleanliness, but a small additional amount of marketable lead-concentrate.

It became quite clear, however, that the Elmore process was not capable of treating slime satisfactorily, either alone or with sand. Slime treatment, however, the process suggested by Sulman, Picard, and Ballot, in conjunction with an apparatus designed by T. J. Hoover, was able to accomplish. In 1910, therefore, this latter process—into which the suggestions of Greenway, Higgins, Lavers, Hoover, and others, had been brought under the aegis of the Minerals Separation Company—displaced the Elmore process, proving itself competent to treat sand and slime alike; its initial success was, in fact, with slime. From that year onward, the problem of recovering the zinc at Broken Hill was solved, and hundreds of thousands of tons of marketable zinc-concentrate became recovered annually.

With this spectacular success to its credit, flotation extended rapidly. At the Kyloe mine in New South Wales, the Minerals-Separation process was applied in 1911 to the recovery of chalcopyrite from an ore containing much less sulphide mineral than at Broken Hill, the content of copper being about 5 per cent.[1] Following this came its introduction into the United States at the Butte and Superior mine, Montana, in the year 1912, a modified Minerals-Separation process there achieving a remarkable improvement in the recovery of fine zinc-blende. Probably, however, the greatest forward impulse which flotation received was the successful application of the Minerals-Separation process to the beneficiation of the immense disseminated-copper deposits of the United States and elsewhere, this success, after some experimentation, being complete early in 1914. From that year onwards flotation has been a matter of the greatest interest upon all mineral fields throughout the world.

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[1] Ashcroft, *Trans. I.M.M.* vol. xxii., 1913, p. 3.

Flotation Processes.—Considering only those which have been commercially applied, the foregoing recital has disclosed the following processes :—

I. Defined by the nature of the lift or buoyancy.
    Oil lift (Elmore Bulk-Oil).
    Gas-bubble lift (Potter-Delprat).
    Relative lift (De Bavay).
    Mechanical lift (Macquisten).
    Air-bubble lift, with limited air (Elmore-Vacuum).
    Air-bubble lift (Minerals-Separation).

II. Defined by the nature of the float.
    Oil flotation (Elmore Bulk-Oil).
    Gas-froth flotation (Potter-Delprat).
    Film or skin flotation (De Bavay, Macquisten).
    Air-froth flotation (Elmore-Vacuum, Minerals-Separation).

Present-day flotation is air-bubble or air-froth flotation, commonly described as ‘froth flotation.’

FROTH FLOTATION

From the foregoing recital it is apparent or foreshadowed that flotation depends upon or is related to the following physical factors :—

I. The non-wetting property of metalliferous sulphides. This non-wetting resists the spread of water over the mineral surface, that is, resists the displacement of air by water. In flotation, non-wetting manifests itself by the attachment of particular minerals to air-bubbles in water.

II. The affinity of metalliferous sulphides for oil particularly, but also for other contaminants.

III. The property of certain insoluble contaminants, and particularly of oil, to increase non-wetting.

IV. The property of certain soluble contaminants, and particularly of soluble oils, oil-fractions, alcohols, esters, turpenes, etc., to secure the maintenance of fine bubbles in water.

V. The property of attached mineral-particles and of contaminants to strengthen the bubble, permitting a froth to form at the surface.

VI. The property of certain agents—acids, alkalis, and salts—to modify the above properties, making them more positive or selective.

It is evident also that mineral flotation is carried into effect by the following series of operations :—

1. Reduction of the material to the necessary small size, and the preparation of the aqueous pulp.

Flotation is not successful on particles larger than about 40 mesh ; moreover its particular province is to recover such fine mineral-particles as for their release require the material to be ground to 80—120 mesh or finer, such particles being readily floated. Flotation also requires a definite mobility of the particles in the aqueous pulp in which they are borne, experience indicating that with sandy material a liquid-solid ratio of 3·5 : 1 and with slime a ratio of 7 : 1 represent the order of suitable dilution.

Reduction in size and preparation of pulp are accomplished in the wet crushing of the ore.

2. Mixing with the contaminating agents.

3. Introduction and sub-division of the air, that is, aeration.

4. Collection and handling of the resultant froth.

These last three operations are conducted in the flotation plant proper.

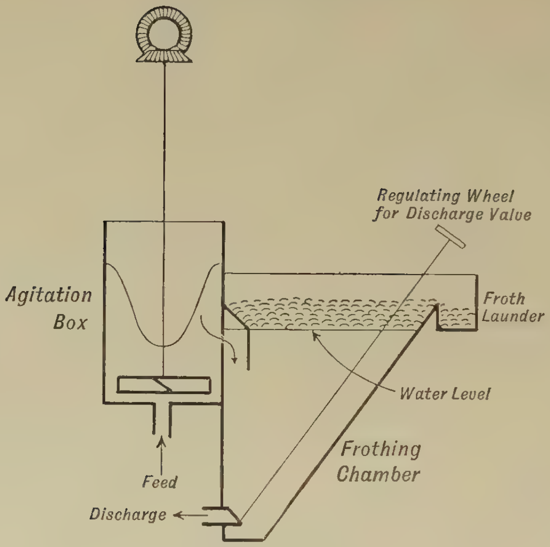
Flotation Machines

The apparatus in which air is introduced and sub-divided, and the mineral-laden froth is formed and collected, is the flotation machine ; mixing with the contaminants takes place either in this machine or in a separate apparatus previously.

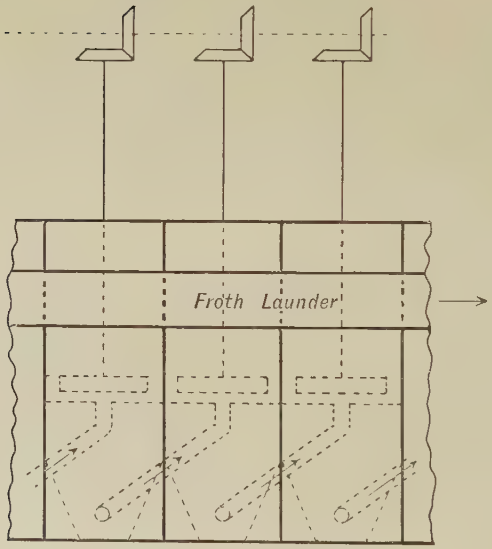
Of flotation machines there are three main types : the first, that in which mixing and aeration take place together and by means of a fast revolving impeller, these being known as ‘ mechanical ’ machines ; the second, in which the fine division of the air is accomplished by forcing air through a porous bottom, the mixing having taken place previously in a separate vessel, these being ‘ pneumatic ’ machines ; and the third, a machine wherein, after separate mixing, aeration takes place by the entrainment of air as the pulp falls from one separating vessel to another, these being ‘ cascade ’ machines.

Mechanical Machines.—Of the above three types the mechanical flotation machine is the widest applied ; it was also the earliest introduced. Violent agitation was one of the specified features of the original Minerals-Separation process, and the machine associated with that process may be taken to be the typical mechanical machine.

The Minerals-Separation standard machine is one constituted of several



Cross Section

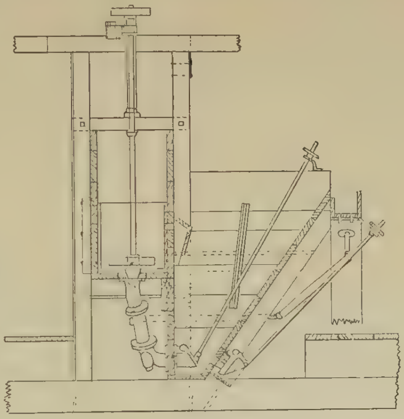


Front Elevation

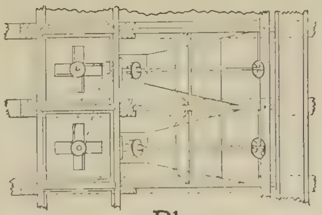
Fig. 291.

Minerals-Separation Standard Cell.—Diagrammatic Elevations. The discharge from one cell becomes the feed for the next, the pulp passing from the frothing chamber of one cell to the agitation box of the next. This cell, and particularly the disposition of the frothing spitzkasten in front of and directly attached to the agitation box, was largely the design of T. J. Hoover (p. 399).

cells in series, each cell consisting of a square agitation box 18—42 in. in section and 24—60 in. in height, standing on end, open at the top, and having in front a frothing chamber, rectangular in plan but triangular in side elevation (Figs. 291, 292). Though these two compartments are contained between the same side-planes and form one piece, they are not on a level, the agitation box extending above the frothing chamber, the frothing chamber coming to a point below the level of the agitation box, the connection between the two being a slit about half-way up the partition common to them both.



Section



Plan

Fig. 292.

Minerals-separation Standard Cell.—Section and Plan. The vertical spindle is seen to be belt-driven; in other designs it is driven by gearing; in others again when the boxes are very large, each spindle is driven by its own motor. The steep baffle in the frothing chamber delays a too rapid arrival at the overflow lip. The suction connection between the frothing chamber of a preceding cell and the agitation box of a succeeding cell is seen in both section and plan. The frothing chamber inclines inwards not only from the front, but from the sides, so that it is roughly an inverted pyramid (p. 401). (Ashcroft, *Trans. I.M.M.* Vol. XXII., 1913, p. 12.)

At the bottom of the agitation box is a horizontal paddle or impeller on a vertical spindle driven from above. This impeller, which consists of four inclined blades, and is from 12—14 in. diameter, is rotated 250—350 revolutions per minute, or, say, at an average peripheral speed of 1500 feet, in a direction to cause suction below, drawing the pulp through a pipe centrally penetrating the bottom. This pulp it throws to the periphery and up the side, leaving a vortex depression at the centre. At the same time and in the turmoil, over the extended and continually-breaking water-surface, air is entrapped, to be dragged within range of the impeller, where it is sheared and re-sheared to an infinite subdivision. This air, it is seen, arrives from above, where, as before stated, the box is open; there also the contaminants and other agents are generally added, if not in one box then in another. Agitation also providing the opportunities for contact, attachment of the sulphide particles to the minute air-bubbles takes place, after which, as more material is sucked in at the bottom, the mixed and aerated material above the impeller finds outlet through the slit leading to the frothing chamber, this slit being about half-way up the vortex column. The mixed material thus thrown into the frothing chamber is not projected directly across but deflected down by a suitably-disposed baffle, around which the mineral-laden bubbles must pass before they begin their flotative rise to the surface; the gangue particles in the meantime continue their descent to the bottom, where, through a regulating valve, they are discharged.

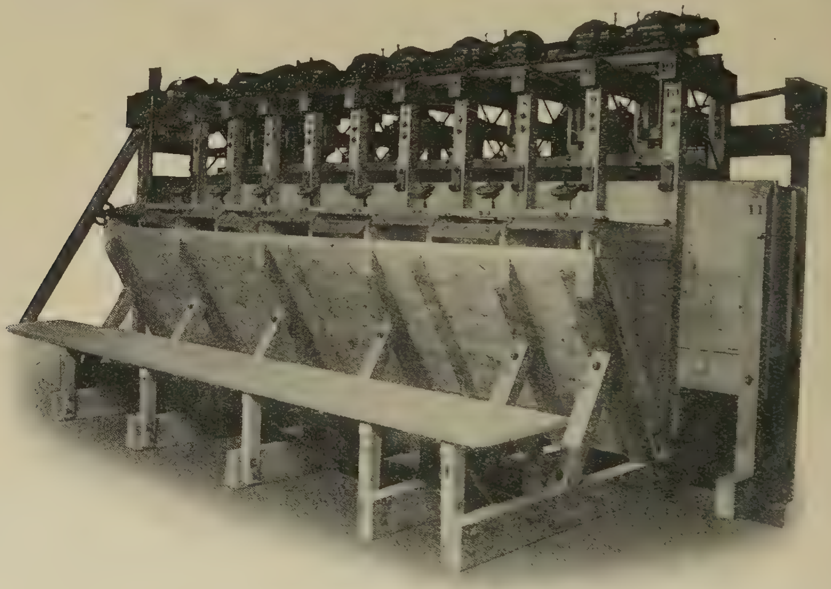


FIG. 293.

Minerals-separation Gear-driven Machine.—General View. Shows hand-wheels for adjusting the regulator-valves; also the skimmers, these being driven from the motor shafting at the far end (p. 403).

Arrived at the surface the minute bubbles coalesce to form a larger-bubbled froth, the depth of which depends upon the height of the overflow lip and the level at which the water is maintained. By adjustment of the regulating valve this water-level is ordinarily 3 or 4 in. below the lip over which the concentrate is discharged, and a little above the slit through which the mixed material enters; accordingly, the depth of froth is not great. As a safeguard against the overflow of water with the froth at fluctuations of feed, a weir is often provided at the back of the frothing chamber over which excess water may drop to the tailing discharge. With no flow of water to assist discharge, it is usual to remove the froth by a mechanical skimmer, generally a sheet-iron paddle revolving slowly over the lip, cutting into the froth at each revolution (Fig. 293); where, however, the froth is abundant it may build high enough to effect its own discharge.

The complete Minerals-separation machine consists of a number of such cells in series (Figs. 293, 294). Into the first of these the pulp arrives at a proper consistency of about $3\frac{1}{2}$ of water to 1 of solids when sand is being treated, and about double that amount of water with slime. This pulp pours into the agitation box above the impeller, is mixed and passes thence into the frothing chamber, whence after flotation of the bubbles it becomes the feed to the second agitation box; and thus down through the series. The length of the series depends upon the material being treated; formerly eight cells were considered to constitute a standard machine, but to-day it is more common to see almost double that number. With such a large

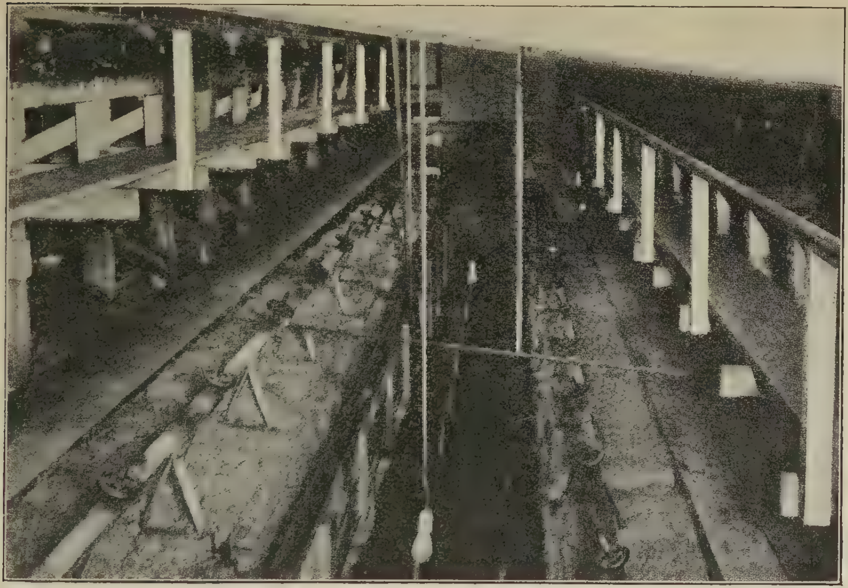


FIG. 294.

Minerals-separation Flotation Machine.—General View at Anaconda. Two machines, each of many cells, are seen with a service way between them. The froth is seen to be voluminous and to need no skimmer. The wheels for adjusting the discharge valve are prominent (p. 403). (*M. & S.P.*, August 28, 1915.)

number the first cell often has no frothing chamber, but is used entirely for mixing, aeration, and the more complete emulsification of the oil. So also with the remaining cells, they may not all have the same function; flotation being at best a differential process, the more floatable minerals, monopolizing the froth, rise as a relatively clean concentrate in the first cells, while from the later cells a rough concentrate or middling may be collected, leaving a tailing to flow away; such a middling product would be returned to the head of the machine to be cleaned (Fig. 295). The more usual practice, however, is to send the froth from all the cells as a single product to a second machine of smaller size and having a smaller number of cells, this second machine making a clean concentrate and a middling returnable to the first machine. Two machines so dividing the work would stand to one another in the relation of rougher and cleaner, respectively (Fig. 295). In returning middling to be cleaned care must be taken that the amount of this product is not excessive or it will accumulate in the circuit.

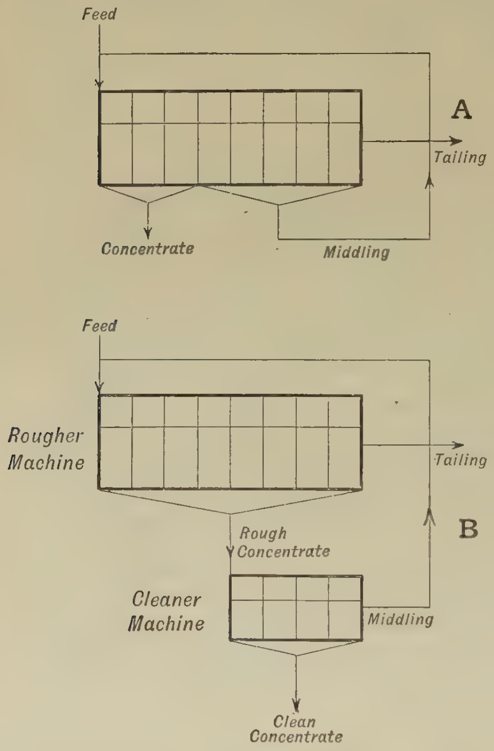


Fig. 295.

Minerals-separation Flotation Machine.—Diagrams. A, showing the different products which may be drawn off a single machine; and B, the products around two machines standing as rougher and cleaner to one another (p. 404).

These Minerals-separation machines are undoubtedly efficient. The violent agitation permits the use of thick, unrefined, and consequently cheap oils, these oils becoming satisfactorily emulsified. In addition to being cheap, such oils also help to form a tough and coherent froth, wherein relatively heavy and large pieces of mineral may be borne and thereby recovered. This good point of the machine, however, is purchased at the expense of considerable power; the impeller of a cell of average dimensions will require about 5 h.p., and that of the largest machine as much as 10 or 12 h.p. Where, however, the material is difficult to treat, that is to say, where it contains but little mineral or itself is very fine, no other machine appears to give quite such good results. It may be said generally that it has a capacity of about 1·25 tons per square foot of machine area per day, including the area of the agitation compartments; and that it consumes about 0·25 h.p. per ton treated per day, or, say, about 4·5 k.w. hour per ton of ore. A machine of ordinary size will treat about 400 tons per day and require about 100 h.p. The division into separate cells ensures continual retreatment; it permits the line between concentrate and middling to be varied to meet changing conditions; and allows the addition of supplementary amounts of flotation agents down the series. In the standard design it is also possible to adjust the water-level of each cell to its own requirements, by altering the opening of the valve at the bottom, and of this adjustment use is still made in many plants. Recent tendency, however, is to preserve a constant level throughout the series, the valve at the bottom falling out of use as a regulator. Such a constancy of level is obtained by open gates between the frothing chambers, these gates allowing at the same time a greater circulation of the pulp, some of the material returning upstream through them.

One type of machine, known as the ‘valveless’ machine, has neither the valves nor the suction pipes from cell to cell. In this type the agitation box and the frothing chamber are on a level, the partition between them determining a circulation into the agitation compartment from below, and into the frothing compartment from above. Progression down the series is obtained by so staggering the frothing chambers in front of the agitation boxes that the impeller draws its feed directly from the frothing chamber of one cell to throw it directly into that of the next (Fig. 296). This type of cell, while requiring less height, takes more power, since the impeller works under a greater depth of water.

Another mechanical machine, the Janney, has a frothing chamber on either side of a central agitation-compartment, wherein two impellers on the same spindle are directly driven by a motor above, sucking pulp equally from the bottoms of the two frothing chambers, and throwing it into them again from on top (Fig. 297). In a complete machine of this type the pulp flows directly from frothing chamber to frothing chamber, the chances being that it is caught by each suction in turn.

It has been argued that to introduce and divide the air by a revolving impeller is unduly expensive in power, because the vortex has to be maintained, and because, in the absence of any regular circulation of the material through the mixing zone, if all the material is to be sufficiently mixed much will be over-mixed. Besides, the entry of the air from on top across a troubled water surface necessitates a separate chamber where flotation may take place under the quiet conditions which it demands. These considerations have led to modifications.

In the Minerals-separation 'sub-aeration' cell the agitation box is itself, in its upper portions, the frothing chamber, the necessary quiet conditions being secured by introducing the air through a pipe far below, and by preventing the vortex by baffles projecting inwards from the box walls (Fig. 298). Above these baffles comes the quiet zone, wherein the loaded bubbles float upward to the surface, forming there a froth, while the gangue particles, rising with the stream sucked in from below, escape through a conduit above the impeller. To lessen the chance of any gangue particles by hazard being caught within the froth, and at the same time

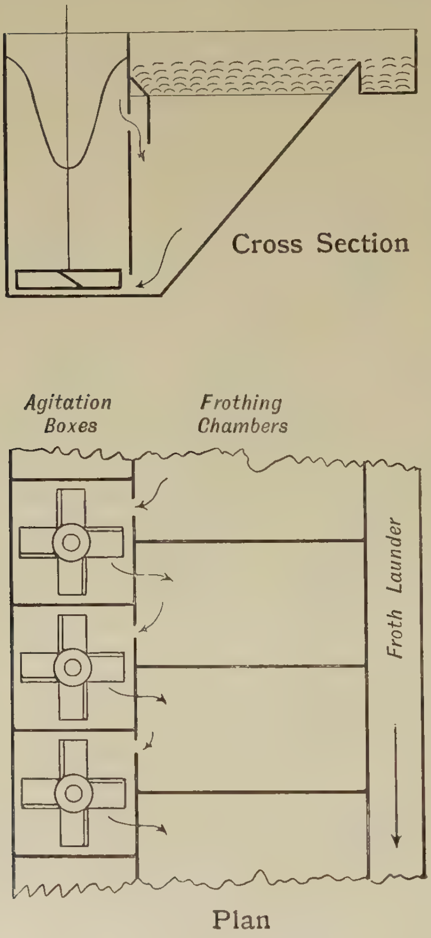


Fig. 296.

Minerals-separation Valveless Cell.—Diagrammatic Section and Plan. In this design the agitation box and the frothing chamber are on a level, and the connection between the chamber of one cell and the box of the next is not therefore through a pipe passing centrally up through the box bottom, but through a slit at the bottom of the partition, the chambers being so staggered in front of the boxes that pulp is sucked into an agitation box from the preceding frothing chamber at the bottom, and delivered through an upper slit into the succeeding chamber. In a complete machine of this type the water level is the same throughout, while, as stated above, the two compartments of a cell are on a level; the machine is accordingly sometimes described as the 'single-level' machine. It has the advantage of requiring less height than the standard type, but, the agitation taking place in deeper water, more power is required (p. 405).

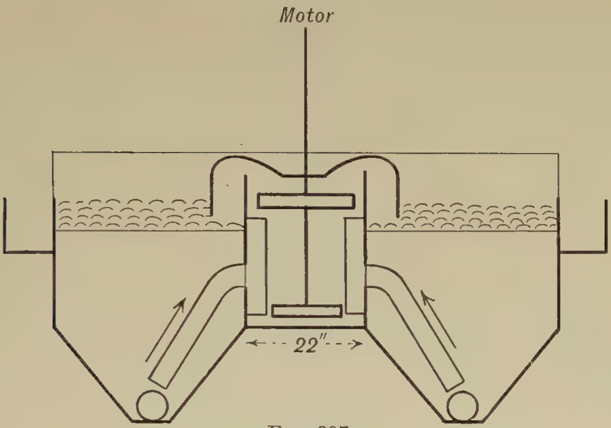


FIG. 297.

Janney Flotation Cell.—Diagram. This cell has two disc impellers on a direct motor-driven spindle working in an agitation box between two frothing chambers, one on either side ; it represents the double-spitz type of cell. These cells are grouped in series, succeeding cells being six inches lower than those preceding. The pulp passes from the frothing chamber of one cell to the frothing chamber of the next, and not from frothing chamber of the upper cell to the agitation box of the next. This machine was developed and used at the Utah Copper (p. 405).

to make the froth more compact, the upper portion of this quiet zone is constricted by inclined walls, the frothing area being thereby diminished. At the top comes a launder into which, over a discharge lip, the froth concentrate falls; no skimming device is necessary, the volume of the froth and the excess of air ensuring discharge. This cell was one of the earlier forms; in practice the position of the tailing discharge was not entirely satisfactory.

Another sub-aeration machine maintains the front frothing-chamber, but raises its level so that it continues above the agitation box, an inclined cover over the latter leading the pulp away from the mixing zone, upwards into a conical hood over the frothing chamber, where the froth forms, and is discharged (Fig. 299); it may be also, that instead of an impeller with inclined blades a simple disc will

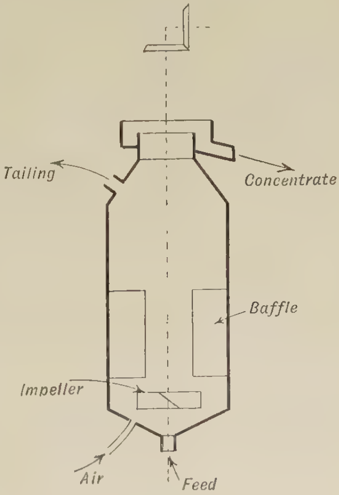


FIG. 298.

Minerals-separation Sub-aeration Cell.—Diagram of the Owen Cell (p. 406).

suffice, or two discs upon the same spindle, with fixed baffles both between and above. More recent designs of the Minerals-separation sub-aeration machine have, however, returned to the idea of a single compartment; there is also the tendency to make the individual cells less definite by placing, for instance, a number of impellers in a trough, the cells being roughly separated by baffles, and the agitation zone below divided from the quiet frothing zone above, by a wooden grid, through the openings of which the loaded bubbles have opportunity to rise.

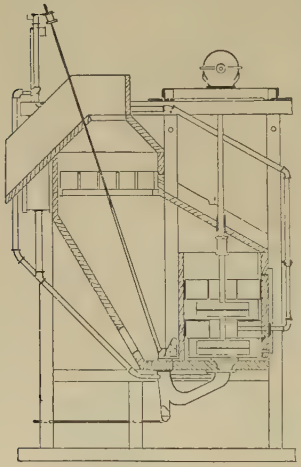


Fig. 299.

Minerals-separation Sub-aeration Cell.—Cross-section of the Hebbard cell tried at Inspiration. The air service is seen to include entry into the suction and into the agitation box. Two discs on the same vertical spindle effect the mixing of the material and the fine division of the air. It is seen that this spindle is driven by gearing and not by belting (p. 407). (Gahl, *Trans. A.I.M.E.* Vol. LV., 1917, p. 592.)

Sub-aeration as applied to mechanical flotation machines does not appear to have demonstrated itself the equal of the ordinary means of aeration from above, for general work. Including the power necessary to deliver the air at the pressure necessary, about 5 lb., it may consume less power, the discs not lifting so much as inclined blades; it also allows control of the air and thus makes the flotation of one mineral in preference to another more practicable; since its success depends upon volume of froth rather than upon strength of froth, it may also require less insoluble oil and therefore be more suitable for treating slime, this material giving an unselective float with much insoluble oil. These points are dealt with more fully in connection with pneumatic machines.

Other modifications of the typical mechanical flotation machine have appeared under other names. The Ruth machine was designed to meet the presumed objection that the vortex formation was wasteful of power, and to comply with the idea that the necessary contact between air-bubble and sulphide-particle was best obtained when both were proceeding in the same direction and approximately at the same rate. Accordingly, the impeller is not open but shrouded, not upon a solid spindle but upon one which, being hollow, permits air to be taken in at the upper end and pulp at the lower (Fig. 300). From four hooded-ports in the upper shroud of this impeller, the air and pulp are thrown upwards together, the mineral-laden bubbles continuing upward to become froth, the gangue particles gradually falling under the action of gravity till, bereft of their forward impulse, they are directed backward by the sloping front of the box, to come again under the lower end of the spindle, there to begin the cycle afresh. It is claimed that after passage through the definite mixing zone the then oiled and aerated pulp passes to the floating zone, where the mineral bubbles remain, leaving only that portion which requires further mixing to pass into the mixing zone again, power being thereby saved. It is said that the impeller may run at a slower speed than in the Minerals-separation machine, and, since the water surface is not broken, froth may form above the impeller, permitting a more compact machine.

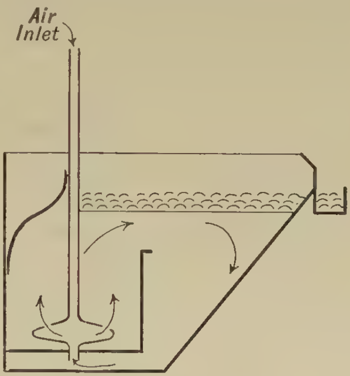


Fig. 300.

Ruth Flotation Cell.—Diagrammatic Section (p. 408).

Of very similar design is the Groch machine, which likewise has a shrouded impeller upon a hollow spindle, air and agents being taken in above, and the pulp, after passing to the frothing zone, being returned to enter beneath. The design differs, however, in that the pulp is thrown out to either side, a number of impellers being placed regularly down the middle of a V-shaped box about 14 ft. long and 5 ft. wide at the top, this box being divided both longitudinally and transversely to demarcate the separate cells (Fig. 301).

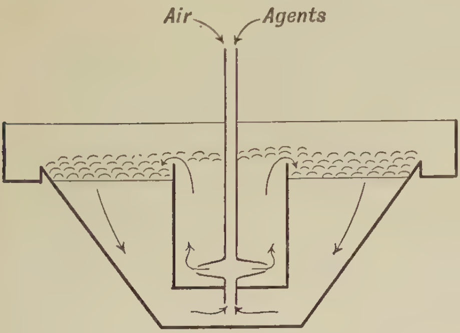


Fig. 301.

Groch Flotation Cell.—Diagrammatic Section (p. 409).

The machines so far described have impellers revolving round vertical axes; others, as at Bunker Hill, circulate the pulp from cell to cell by centrifugal pumps, each sucking air through a snifting valve; others, again, have the simple impellers merged into a drum revolving round a horizontal axis extending from end to end of the machine. Of this last type, the K-and-K machine has a drum or rotor which consists of four steel spiders upon a stout shaft, these spiders carrying longitudinal lagging spaced about a quarter of an inch apart round the circle (Fig. 302). On the outside of the lagging are hardwood riffles running the entire length. This rotor is contained in a box about 12 ft. long and 30 in. square, within which it revolves at a speed of about 180 r.p.m., and with a clearance of about $\frac{3}{8}$ in. The shaft on which it is mounted passes through two air ducts, one in each end-plate of the machine, through which air is sucked to aerate the pulp as it is carried round by the riffled surface. In front of this agitation box is a sloping frothing-chamber, communication between the two being through holes situated low down in the partition. The pulp enters one at the end of the machine and is discharged at the other, taking a helical path upwards in the agitation box and downwards in the frothing chamber, the mineral froth being discharged over a lip in front, the pulp returning to the agitation box. Discharge of the tailing takes place eventually through a pipe which, emerging from below, rises up to about the water level and then descends again, acting as a syphon. It is seen that there are no separate cells; the power required is small, being only about 10 h.p. for the size of machine mentioned, this being stated to have a

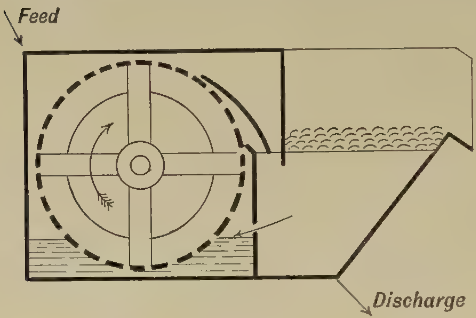


FIG. 302.

K-and-K Flotation Machine.—Diagrammatic Section (p. 409).

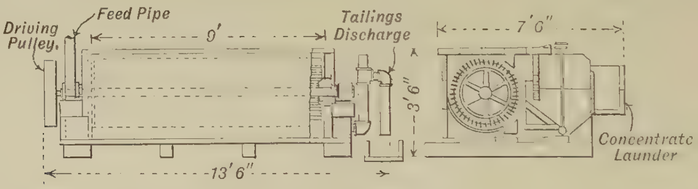


FIG. 303.

Rork Flotation Machine.—Cross-section and Side Elevation. This machine consists of a wooden casing or chamber, fitted inside with baffles and diversion plates, and enclosing a hollow drum with fan blades at the periphery; this drum revolves with a minimum of clearance. In front is a frothing chamber with sloping bottom; at one end is the feed pipe, and at the other end a syphon discharge. The frothing chamber is divided transversely into three compartments, in each of which controllable valves at the bottom determine the precise height of the water level. A machine of the dimensions illustrated will treat 35—40 tons of 80 mesh material per 24 hours, and require about 5 horse-power (p. 411). (Westby, *E. & M.J.*, February 24, 1917, p. 336.)

capacity of 80—120 tons per day; the floor space is only about 4 ft. by 14 ft., and the head-room only about 3 ft. The Rork machine is similar (Fig. 303).

Another horizontal machine consists simply of a number of discs upon a rapidly-revolving shaft carried upon the end-plates and partitions of a V-shaped box. These discs being part submerged in pulp carry air down on one side and bring up aerated pulp on the other, with so little disturbance that froth forms on the surface. From feed to discharge the pulp passes through holes in the partitions, concentric with the shaft. In general, horizontal machines appear to do good work with slime.

Pneumatic Machines.—The principal and prior type of pneumatic machine is the Callow machine, which consists of a rectangular box about

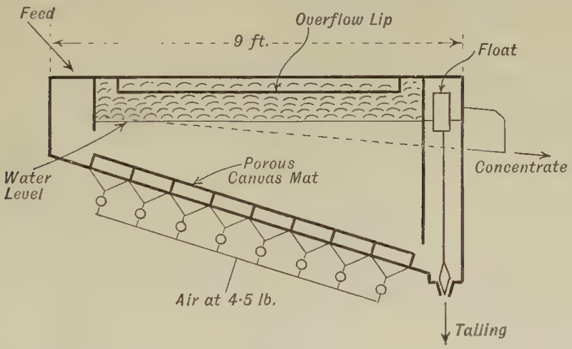


Fig. 304.

Callow Pneumatic Cell.—Diagrammatic Section. The float automatically controlling the discharge is for clearness shown in a separate compartment; in reality, the float is situated in the froth, its movement actuating the end of a horizontal lever having its fulcrum hinged in the end plate; to this lever the valve spindle is connected not far from the fulcrum (p. 411).

2 ft. wide and 8 ft. long, with a sloping bottom, the depth being about 20 in. at one end and 45 in. at the other (Fig. 304). This sloping bottom is not solid but of a porous material, generally of closely-stitched canvas held between wire netting. This porous bottom is the cover to an air box into which a blower delivers air at about 5 lb. pressure. In consequence of greater depth, the hydraulic head under which the air issues at the deep end is greater than at the shallow end. To equalize the issue of the air under these conditions, the air box is divided into compartments each separately served from the air main, the amount of air passing into each being regulated by a separate valve. The shallow end of the box is the feed end; upon entry the feed is directed downward to the air mat by a baffle, passing below which it meets the

flood of rising bubbles, the sulphides attaching themselves and rising to form a voluminous froth, the gangue particles continuing down the slope to pass out at the deep end through an automatically-controlled discharge. The froth so formed, building itself up from below, eventually overflows a discharge lip running the greater portion of one side, falling into a launder (Fig. 305). In this machine the water-level and the depth of froth are matters of more than ordinary moment. The bubbles which rise to the underside of the froth always carry some gangue; being relatively weak and brittle many break, but the rate of arrival being greater than the rate of breakage the froth as a whole rises. At each break a load is shed, the mineral particles attaching themselves to other bubbles and rising again, the gangue particles losing hold; accordingly the higher the bubble the cleaner its load. To bring about the desired cleanliness of concentrate the water-level is usually maintained some 12 to 14 in. below the discharge lip. Constancy of this level throughout fluctuations of feed is obtained by the automatically-controlled discharge, namely, a float upon the water surface, which, as it rises and falls, enlarges and contracts the discharge aperture, respectively.

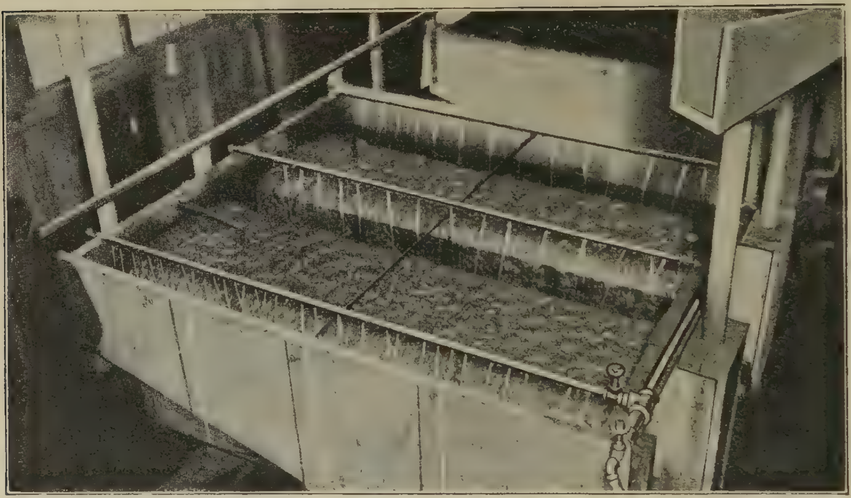


FIG. 305.

Callow Pneumatic Cells.—Overhead View, in operation. The feed enters by the vertical pipes on the right, the tailing discharge is on the left; the horizontal lever of the discharge control and the adjusting nut on the valve spindle are seen in the near cell. Froth discharge takes place on both sides of the cell; the launders which in this installation carry it away have an opposite inclination to that of the box-bottom. The sprays employed to break down the froth in these launders are clearly seen (pp. 412, 453). (*M. & S.P.*, August 28, 1915.)

It is seen that with this machine the fine division of the air is accomplished by passage through a porous medium and not by violent agitation. The amount of air so required is generally about 10 cub. ft. per minute per square foot of porous area, but, its pressure being low, the power consumed is substantially less than would be required for an equivalent mechanical aeration ; with sandy material a standard cell of the dimensions given would have a capacity of about 50 tons per day, at which figure the power consumed would be about 0·15 h.p. per ton treated per day, or, say, 2·7 k.w. hour per ton. But seeing that with this aeration no adequate mixing of the oiling and frothing agents takes place, such mixing must be done previously. In treating sand, and where no acid is employed, the oil may be added ahead of the grinding machine, that machine at no extra expense then serving as an excellent mixer. Where, however, acid is essential, mixing must be undertaken in a special mixer ; a special mixer is also necessary for slime since this material needs no regrinding. An appliance commonly used for this purpose is the Brown agitator, a tall tank with a pipe standing centrally within, up which pipe compressed air entered at the bottom lifts the pulp, discharging it at the top, whence it falls to begin the cycle again (Fig. 306). A tank suitable to the capacity of a standard cell would be about 18 ft. high and 4 ft. diameter.

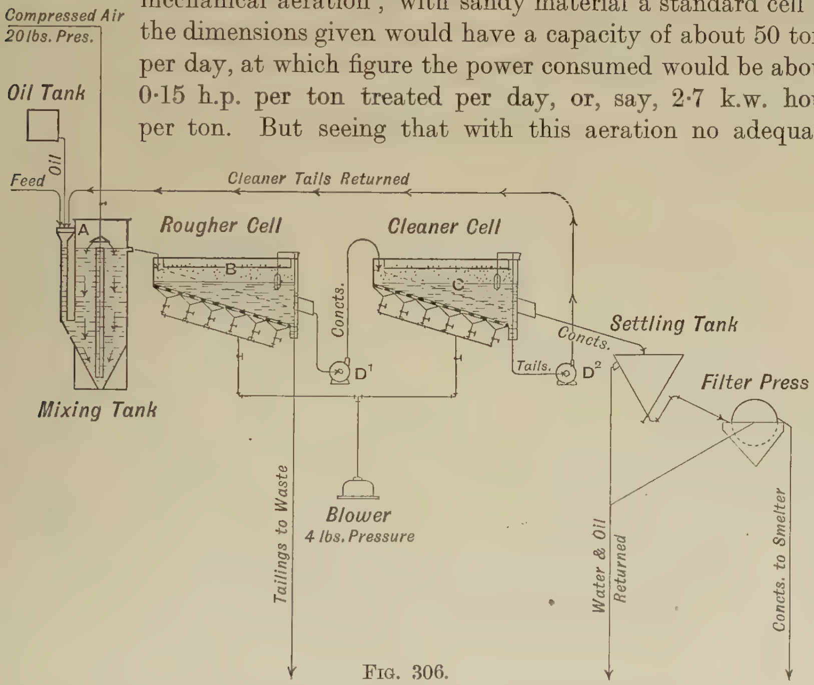


Fig. 306.

Callow Pneumatic-flotation Installation.—Flow-sheet. Diagrammatic representation, showing Mixing, Floating, Cleaning, and Concentrate-Handling, in proper sequence (p. 413). (*Trans. A.I.M.E.* Vol. LIV., 1917, p. 6.)

Including the air necessary for such previous mixing, experience shows that the power consumption of the Callow machine approaches that of impeller machines.

The figures of capacity for sandy material given above are equivalent to about 3 tons per square foot of porous area per day, or, say, about 2 tons per square foot of total machine area. With slime the capacity would be much lower, and generally less than 1 ton per square foot of total area per day.

It is seen that the Callow machine is not an assembly of cells, but a single cell; it accordingly produces but two products, that carried over with the froth and that discharged as tailing; normally no middling product can be taken. Treatment in a single cell is, however, rarely sufficient. Generally, by reason of the extensive frothing area, the material which finds place in the froth from a first treatment is not clean enough, only one final product being made, namely, a clean tailing. The enriched but not yet clean concentrate is retreated in a second cell, which acts as a cleaner to the first (Fig. 307). This cleaner cell has usually the same dimensions as the first, the cleaning effect arising from the fall of most of the remaining gangue at this second opportunity. The water-level may be maintained lower and the depth of froth greater, the mineral then reaching the discharge lip being that which has survived the hazards of a relatively long journey upwards; a modifying agent may also perhaps be added to increase the wetting power of the water; usually, no further oil is added. All these attentions

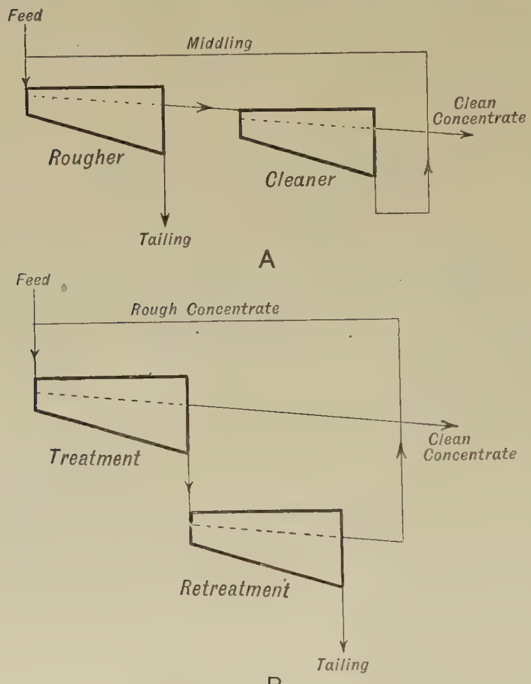


Fig. 307.

Pneumatic Flotation in Machines Series.—Diagrams. A, Roughing Series, in which the tailing is discarded after a single treatment in the rougher machine. B, Retreatment Series, in which tailing is discharged after two treatments and only the retreatment concentrate is cleaned (pp. 414, 415, 647).

so minimize any tendency of the gangue to float that the concentrate is a finished product; the tailing, on the other hand, generally contains sufficient mineral to warrant its return to the head of the rougher machine, being in effect a middling product.

If, however, the conditions are so set that the first treatment gives a finished concentrate, it will be likely that the tailing must be retreated. Operating in this way and returning only the rough concentrate from the second treatment to be cleaned in the first machine, the advantage of a double treatment before discard of the final tailing is obtained (Fig. 307). This 'retreatment method' is advocated where no other second treatment of the tailing is provided; but the 'roughing method' is more common.

It may here be remarked that the Callow machine by reason of the great depth of froth which is one of its features makes a good cleaning machine. More than that, it can be made to discharge one mineral in preference to another. Operating on mixed lead-zinc sulphides, it is quite possible so to adjust the depth that only the more floatable galena passes over as concentrate, the less floatable blende not surviving. In the standard design, as already mentioned, the concentrate discharge is over one side; but if it be over the deep end, or again, be over the two sides but near the feed end, the froth rising near the particular closed end must travel a longer distance to reach discharge, and may, indeed, be made to pass over transverse baffles on the way. Such exceptional arrangements increase the cleaning or selective action of the machine.

Modifications of the Callow machine have developed shallower depth and a division into cells. The depth at the feed end has been lowered to 15 in., and that at the discharge end, after a long intervening length of 16 ft., to about 24 in. The machine then becomes almost a launder machine, the pulp being carried forward chiefly by stream action, the inclination of the bottom being too small to be of any great assistance; in this progression the bubbling of the air assists by keeping the sand mobile. Under such a shallow depth the air-pressure required is only about 3½ lb. per square inch.

The division of the pneumatic machine into many separate cells in series was developed at the Inspiration. The Inspiration machine consists of a long box divided into cells by transverse baffles which do not reach quite to the bottom (Fig. 308). In this box the pulp fed at the upper end descends immediately to pass under the first baffle. At this point the porous bottom and the air entry begin, continuing right to the end. Each succeeding cell is similarly constituted; each has a width of about 3 ft. downstream, and 4 ft. across, discharge of the concentrate being on each side into launders running the length. Below the last baffle the tailing rises to the tailing launder, which lies more or less in the same inclined plane as the feed launder, the bottom of the flotation box itself lying at a similarly low gradient about 2 ft. below. The force which keeps this box clear and prevents the sand packing is that arising in the difference in level between entry and discharge, this difference constituting a bursting head. Allowing that the free-ways in the box fill with sand till the accumulating pressure is sufficient to force a passage, this lodgment apparently does not detract from the capacity of the box. Another feature of the Inspiration machine is that the porous mat by which the fine division of the air is obtained, is a separate piece which can be introduced and laid in the bottom from above; moreover, each cell has its own mat, of which the air-box is served by pipes entering from above. This construction, in addition to simplifying the containing box, permits also the porous mat to be removed and thereby readily cleaned or renewed. The entire machine consists of sixteen cells in series, two such machines being laid alongside one another so that when the porous mats of one require attention the whole pulp may for the time pass through the other. Retreatment in series permits this temporary overload without undue loss in the tailing; constancy in the flotation conditions is not disturbed, at least not in respect to the dilution of the pulp, the amounts of agents added, etc. Such a double flotation box has a capacity of about 800 tons per day; it produces a concentrate which is cleaned in a smaller machine. Including this cleaner machine the capacity is about 1·5 tons per square foot of box area per day.

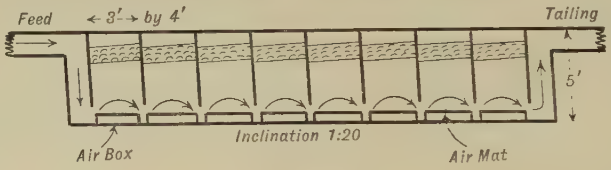


FIG. 308.

Inspiration Pneumatic Box.—Diagrammatic Section (p. 415).
(Gahl, *Trans. A.I.M.E.* Vol. LV., 1917, p. 595.)

In operation, pneumatic machines require some attention by reason of the blinding of the porous bottom, this blinding arising from unavoidable dust in the compressed air and from the precipitation of dissolved salts. The mats accordingly must be regularly cleaned and renewed, renewal taking place about every three months when acid is used, and at intervals of about six months with neutral or alkaline circuits. Other materials than canvas have been tried, stiff porous concrete for instance, but the flexible canvas has proved more satisfactory.

A pneumatic machine of a novel type is the Grondal machine, in which perforated rose-heads of large diameter, one at the bottom of each of a number of separate agitation boxes, take the place of the porous bottom; moreover, no uniform gradient from end to end exists, but use is made of each rising flood of air to carry all the pulp upward and over into a frothing chamber having a bottom inclined in the direction of flow; each chamber is thus made to deliver into the next agitation box (Fig. 309). An interesting point is that since the discharge from the agitation box to the frothing chamber is situated about 3 feet above the air-entry, there is some chance that the sand which inevitably must collect over that entry may be of assistance in subdividing the air.

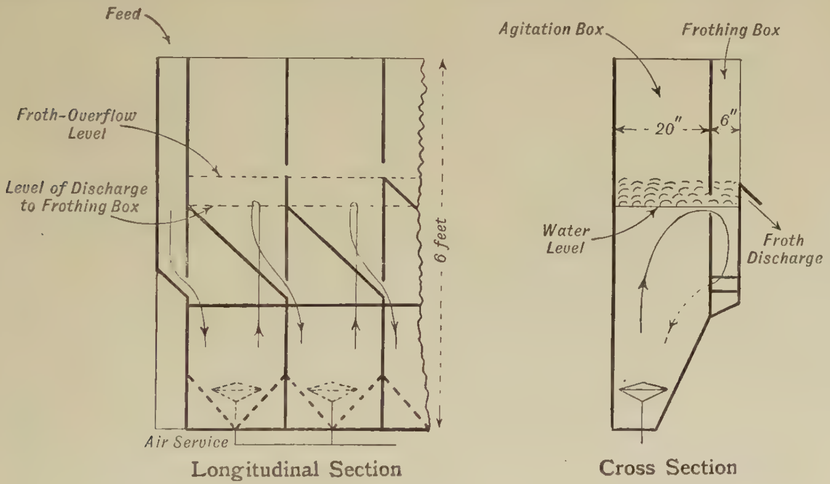


FIG. 309.

Grondal Flotation Machine.—Diagrammatic Sections. Air is introduced at the bottom under a pressure sufficient to overcome the weight of water and crushed ore above. It is divided by passage through perforated rose-heads, one in each agitation box. Eight to ten cells are assembled in series, the capacity of a series of the dimensions given being about 100 tons per day, and the power required about 8 h.p. About one cubic metre of free air is required per cell per minute (p. 417).

Mechanical-pneumatic Machines.—Combined mechanical-pneumatic machines have also been introduced but not been widely adopted (Fig. 310).

Cascade Machines.—Cascade machines make use of the power of a moving stream or jet of one fluid, by friction to carry forward with it an amount of a surrounding or contiguous fluid. This is the principle on which the hydraulic compressor acts, as also do the jet pump and jet ejector ; the principle is also embodied in the working of the steam injector. Accordingly, a stream or jet of water entering quiet water takes air with it, introducing and distributing this air by virtue of its kinetic energy.

The idea of cascade machines was doubtless suggested by the frequently observed formation of froth wherever a stream carrying flotation tailing drops to a lower level. Seale and Shellshear in 1914 at Broken Hill, and others in Arizona and Korea, respectively, were, however, the first to incorporate the idea in a flotation equipment. Since then, though they have not achieved primary importance, such cascade machines have nevertheless occasionally been very helpful.

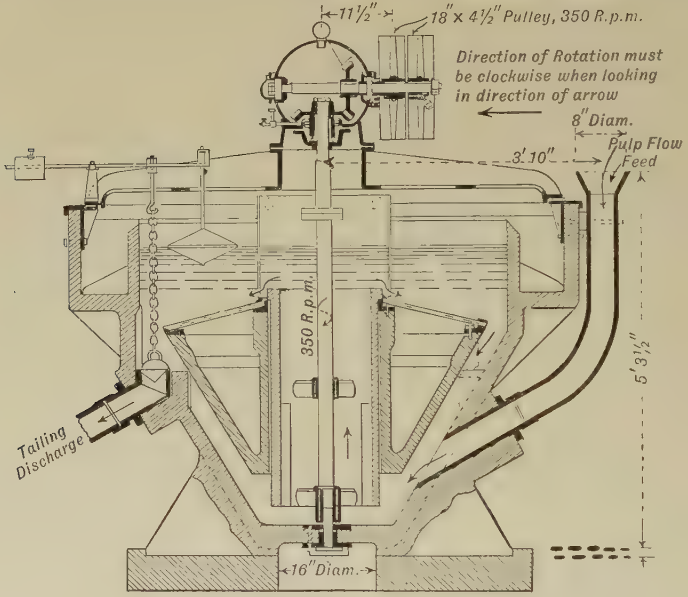

11½"
18" x 4½" Pulley, 350 R.p.m.
Direction of Rotation must be clockwise when looking in direction of arrow
8" Diam.
Pulp Flow
Feed
3' 10"
350 R.p.m.
5' 3½"
Tailing Discharge
16" Diam.

FIG. 310.

Jones-Belmont Flotation Machine.—Cross-section. A central agitation-box, 18" diameter, discharges at the top into a concentric frothing chamber, 5 ft. diameter, through the porous bottom of which air enters finely divided. At the periphery of this frothing chamber the pulp returns to the bottom of the agitator, except for that fractional portion, about one-tenth, which passes out through an automatically-controlled discharge diametrically opposite the feed entry. These cells are used in series (p. 417). (*E. & M.J.*, September 14, 1918, p. 497.)

The entrainment of the air may be effected in various ways all more or less similar. A sheet of water issuing under the requisite head to pass over an apron will drag air with it; experience shows that a flow of

3 or 4 feet down the slope is necessary to effect entrainment sufficient to form a froth upon entry into a quiet pool beneath; in consequence of the difficulty of spreading the stream regularly over the necessary width, this arrangement is not often used.

A second means is to have the water issuing as jets from nozzles in the bottom of a container, such jets passing into receiving pockets directly in line; in crossing the gap between nozzle and pocket, air is entrained, and then, at the pocket itself, actually entrapped. Following the same idea, jets of pressure water or of steam have been suggested. Or, air may be drawn in through induction pipes placed in a larger pipe down which the pulp is falling, the upper ends of these induction pipes projecting above pulp level. The appliance is more simple still if the air is drawn in through holes in the pipe walls (Fig. 311).

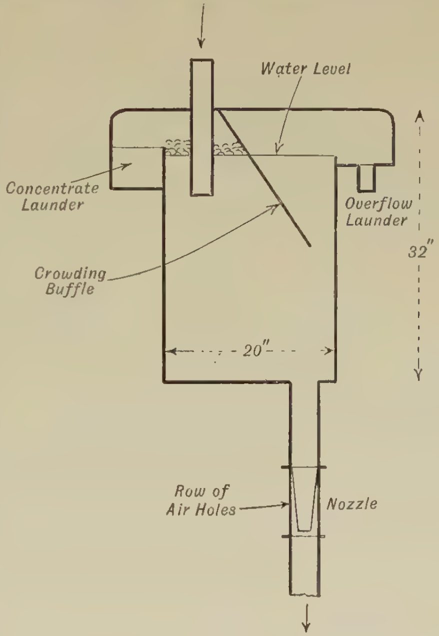
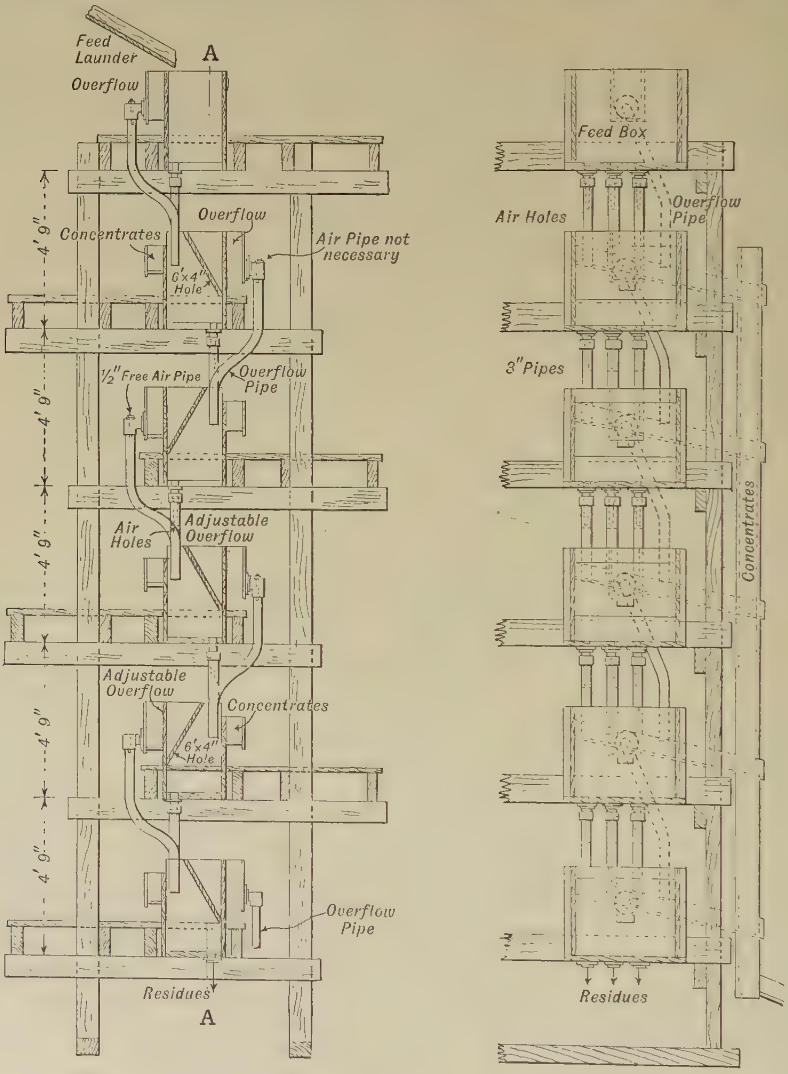


Fig. 311.

Cascade Flotation Cell. — Diagram. Length of cell 36 inches; vertical distance between the bottoms of two consecutive cells 4 ft. 9 in.; three nozzles along the length of each cell; 9 cells in series; intermediate elevation of pulp between the fourth and fifth cells. Two of these units together treated 200 tons per day of reground jig tailings which had formerly been treated by 20 Card tables, making a better recovery and a cleaner concentrate. The tailing from this plant was retreated by differential flotation in a Minerals-Separation plant to recover the fine lead and the zinc (p. 419, Fig. 315). (Harvey, *Trans. I.M.M.*, November 1918.)

A complete cascade equipment consists of a number of frothing boxes stepped one below the other in series (Fig. 312). The pulp, after splashing into the first box and yielding there a portion of its mineral, drops out at the bottom across an air gap to the next, where a second frothing takes place, and so on. The distance between the bottoms of successive boxes is generally about 4½ or 5 feet, of which distance about 3 feet is occupied by the drop, and the remainder by the depth of the box itself. Such retreatment is continued as a rule seven or eight times, usually with one re-elevation of the pulp; the total height required is then not inconveniently great.

Compared with the violent agitation of the mechanical machines and the abundant air-supply of the pneumatic machines, the aeration effected by these simple cascade machines is both mild and limited, with the result that, even granted an adequate previous mixing, the froth is both evanescent and meagre. The frothing boxes, accordingly, are usually provided with



Section A-A

Fig. 312.

Cascade Flotation Machine.—Sectional Elevations of Machine employed at the Central Mine, Broken Hill (p. 419). (*Min. and Eng. Review*, Melbourne.)

an inclined baffle or crowding-board to direct the rising bubbles towards the discharge, and so to compact the froth that the separate bubbles may sustain one another. With such a gentle agitation also, only the readily floatable particles respond, and no complete recovery is made. Seeing its simple construction and low operating-cost, this machine seems particularly suitable for insertion in a departing stream of tailing. It has, however, also been used in advance of the major portion of the flotation equipment; at Broken Hill, for instance, it recovers galena from reground jig-tailing—doing this work better and at a lower cost than tables previously—before the blende is floated in normal mechanical machines (Fig. 315). For its proper functioning it requires a surge or equalizing tank on top of the series to eliminate fluctuations in the amount of pulp. In addition, each cell is provided with an overflow weir.

Machines compared.—Flotation machines must secure the following conditions essential to success:—

1. Mechanical emulsification of the insoluble contaminant, if such be used.
2. Distribution of the soluble contaminant.
3. Entry and minute sub-division of the air.
4. Contact between the contaminated mineral-particles and minute air-bubbles.
5. Ready flotation of the loaded bubbles and quiet formation of froth.

In respect to these conditions, the mechanical machines secure the complete emulsification of the insoluble contaminant, bringing into solution at the same time any soluble fraction associated with it; they also distribute the soluble contaminant rapidly. Thus is mixing achieved. They also accomplish the minute division of the air perfectly, producing in the presence of a proper soluble contaminant such a sparkling effervescence that temporarily the water is rendered milky and opaque (Fig. 313). Though when a lighted match is held over the agitation box the entry of air is at once evident, the amount involved is nevertheless small; accordingly, the bubble efficiency is high. The violent agitation, so successful in dividing the air, multiplies also the opportunities for the essential contact of mineral and air-bubble.

With the sub-aeration type of these mechanical machines the supply of air is more abundant, the froth in consequence more voluminous and the bubble efficiency lower. As with the normal type, the fine subdivision of the air is accomplished by the impeller. With contaminants suitable to each respectively, the normal type, taking in air at the top, produces a less voluminous but stronger froth, capable of retaining relatively large and heavy particles, while the sub-aeration type produces a froth which, both in its abundance and frailty, is more suited to fine material.

With pneumatic machines aeration takes place after a previous mixing in another appliance, which, except the grinding machine be employed, is generally of the pneumatic type, that is to say, of a type unequal to the adequate emulsification of thick insoluble contaminants. The fine subdivision of the air is effected by passage through a porous bottom. Issuing under pressure and over a relative great area the volume of air involved is very large. The froth in consequence is voluminous and at the same time frail, conditions which, as before stated, favour the flotation of fine mineral—it is not an uncommon experience, however, that mechanical machines do cleaner work than pneumatic machines when slime is treated alone. The excess of air finds ready outlet at the free surface of the

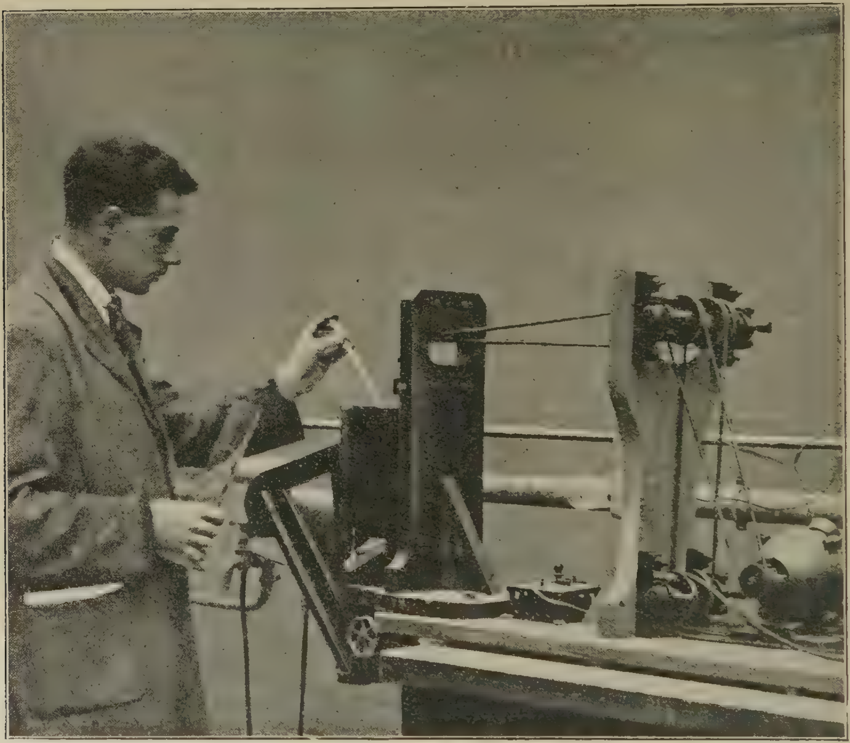


FIG. 313.

Mineral-separation Testing Machine.—General View. The white opacity given to the water when agitation takes place in the presence of an effervescing agent, shows clearly through the glass side of the frothing chamber (p. 421).

froth, where it assists in the discharge, no skimmer being necessary; this excess must not be in such amount that vents form in the froth bed, an occurrence described as 'blubbing.'

Compared with the two previous types, cascade machines are gentle in operation. Like the pneumatic machines they require a previous mixing, aeration being their primary function. Under these moderate conditions, these machines are only equal to raising mineral which is readily floatable.

Mixing, Aeration, and Froth-formation, General

The amount of soluble contaminant necessary is that which under the particular mixing and pulp-dilution will preserve the fine division of the air; when this contaminant is the soluble fraction of an otherwise insoluble oil, the more violent and prolonged the mixing the less oil necessary, since a greater fraction goes into solution. The maintenance of the fine division of the air is the first necessity, since the air, thus reduced to a size with the particles, is the principal selective agent. Beyond this, an amount of insoluble contaminant such as shall increase the selectivity between the air and the mineral particles without agglomerating these latter, is generally advantageous; when this contaminant is the insoluble portion of an otherwise soluble oil, less mixing will be required than were the oil wholly insoluble. Ordinarily the oil or contaminant added in flotation is a mixture, natural or compounded, of soluble and insoluble fractions, the soluble fraction being popularly known as the "frothing agent," the insoluble as the "collecting agent" since it helps to form an elastic and coherent froth facilitating collection.

The amount of such mixture usually necessary to perform both functions is about one-tenth of 1 per cent, *i.e.* 0·1 per cent, of the weight of ore treated, equal to about 2 lb. per ton. The precise amount is primarily dependent upon the amount of water, and particularly of fresh water, present, that is, upon the dilution of the pulp, since water contamination to secure fine air-division or effervescence is the prime necessity; and secondarily upon the amount of mineral present, since contamination of the mineral surface is a necessity only in a secondary sense, or, if regarded as a prime necessity, would largely depend upon the bulk of water in which it was effected. Experience shows that an ore with a high mineral-content, say 40 per cent, does not require much more oil than one containing say 5 per cent. Exceptionally, more than 1 per cent of oil may be used; exceptionally, also, with a readily floatable mineral such as galena and when not much mineral is present, as little as a hundredth of 1 per cent.

Less oil is used, and particularly of the insoluble oil, when the pulp contains much impalpable material, the danger of entraining gangue particles in the froth increasing with the amount of oil; accordingly, when treating slime a minimum of oil is advisable, the best use of this minimum being secured by an adequate mixing. Again, when no modifying agent is used to clean the froth, no acid for instance, it is well to use less oil, achieving cleanliness that way.

With regard to the position at which the oil is added, where acid is not used there is advantage in adding some at least of the oil, and particularly the insoluble oil, in the grinding machine, the newly-formed mineral surfaces then becoming filmed as they form, incipient oxidation being thereby precluded; moreover, the need for subsequent mixing is diminished if not eliminated. This latter consideration is not of such moment with mechanical machines which mix as they aerate, but with pneumatic machines it means the avoidance of special mixers. When acid is used, with mechanical machines it is usual to add the oil and the acid also, in the first agitation box, though smaller amounts may also be added down the series; with pneumatic machines the agents are added in the special mixers, when, if these be worked by compressed air, thick insoluble oils are no longer admissible. The soluble contaminant requires little mixing, but it is usual to add it with the insoluble contaminant, this latter being thereby rendered more mobile. Similarly, the modifying chemicals being soluble require little mixing, but their office being largely to produce conditions favourable to flotation they are in greatest part added early, either with or before the contaminants. There is simplicity in adding the agents once and for all.

If a cleaner machine be in series, it is not usual to re-oil on that account, the purpose of this machine being to give entangled gangue chance to drop out. Moreover, experience shows that both the water and the mineral, once adequately contaminated, sufficiently maintain their contamination. Not only is additional oil unnecessary, it might be harmful. Over-oiling, especially of unemulsified oil, kills froth; the greasy portions form drops which fall out of the froth into the gangue below; the addition of a small amount of free oil is, indeed, one way of breaking down an inconveniently persistent froth.

As already stated, the amount of contaminant is largely determined by the dilution, that is, the amount of water. With dilution kept constant, the rate at which the contaminant is fed must likewise be constant. Such constancy of feed may be accomplished either by special feeders, by drops, or by a running tap. Special feeders often take the form of a small bucket-wheel part-submerged in an oil bath; or that of a simple disc similarly disposed, a scraper applied to its revolving face removing the attached oil-film. Where, however, the quantity of oil used is relatively great, as it would be in a large installation, constancy of feed may be obtained through a running tap, particularly if the oil were thin. Thick oil would be fed by drops, or by a special feeder.

The amount of air introduced is controlled by the duration of agitation with mechanical machines, and by valve adjustment with sub-aeration and pneumatic machines. Its fine division effected, the minute air-bubbles, loaded with their mineral, rise to the surface, where they merge themselves into a more or less stable froth the bubbles of which are very much larger, their actual size depending upon the nature of the oil, the kind of agitation, and the abundance of air. With pneumatic machines a maximum bubble of 2—3 in. is generally preferred, since a smaller bubble does not burst so readily; with the mechanical machines a smaller bubble is more common, though, if desired, bubbles of very large diameter may be formed. Froth is stable largely because its bubbles, unlike those in the water, are formed of two-sided films, such films being elastic and capable of automatically adjusting themselves to changes of physical circumstance. In the matter of its stability froth also owes much to the attached mineral-particles, these, probably more than any adsorbed contaminant, giving it elasticity and strength (Fig. 314);

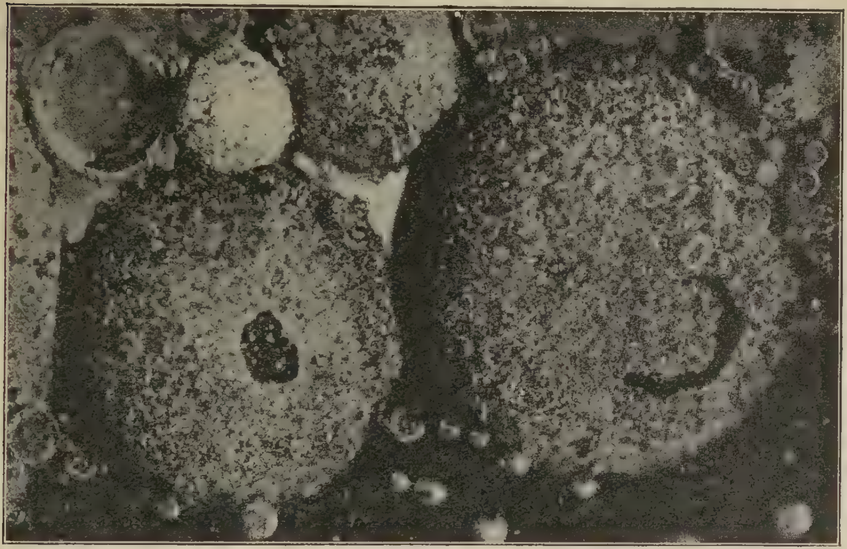


FIG. 314.

Mineral-armoured Air-bubbles.—Micro-photograph; magnification about 100. The four prominent bubbles represent roughly diameters of 12 mesh, 20 mesh, 30 mesh, and 50 mesh, respectively (p. 425). The attached mineral-particles are much smaller. (*M. & S.P.*, May 11, 1918.)

all other things being equal, a well-mineralized froth is more stable than one retaining but little mineral. As stated already, pneumatic machines produce a voluminous and consequently brittle froth, the mineral-particles and contaminant being distributed over a vast total film-surface. Such a froth has difficulty in supporting large or heavy particles. With most mechanical machines, the frothing chamber being separate, the distance to the overflow is greater and a stiffer froth becomes a necessity. Such a froth has, however, the advantage that larger and heavier particles are retained, but on the other hand the manipulative disadvantage of being more difficult to break down.

Flotation Agents and Mixtures

The prime factor in froth flotation undoubtedly is air, the introduction and fine division of which is primarily accomplished by the flotation machine. Assisting air are certain 'agents,' mostly organics and non-electrolytes, and certain 'chemicals,' mostly inorganics and electrolytes.

Of flotation agents, first in importance comes the "effervescing" or 'aeration' agent, which, in solution as a contaminant, promotes the entry and fine division of the air, maintaining this division till the bubbles have risen to merge themselves into a froth. The soluble oils, oil-fractions, wood distillates, tar derivatives are the principal effervescing-agents. Because, in the presence of the mineral-particles of which the air-bubbles have made selection, these effervescing-agents also ensure a copious froth they are described as 'frothing' agents; they are not frothing agents, however, in the sense that soap is a frothing agent with water and saponine with ginger-beer, since without the mineral-particles, and particularly without the fine mineral-particles, no stable froth would generally be formed.

Then comes the "selective agent" which, acting as a contaminant to the mineral, reinforces the tendency of the mineral-particles to attach themselves to the air-bubbles; the insoluble oils, oil fractions, etc., are the principal selective agents. But some of these substances, the tars and thick oils particularly, will, if used, enter the froth to strengthen it of themselves; probably also contaminated mineral ensures a more elastic film than uncontaminated mineral. To this extent the selective agents stabilize the froth. Assisting thus in the collection of the mineral they are sometimes described as 'collecting agents.'

From the foregoing it will be realized that an ordinary commodity might be both an effervescing and a selective agent, effervescing in its soluble portion and selective in its insoluble portion. This happens so often that flotation agents are best described under ordinary terms rather than by their floatative properties. They are, generally: steam distillates and destructive distillates from resinous woods and from hard woods; coal tar and its derivatives; mineral oils and refinery products; fixed oils and fatty acids; and various organic salts.

Distillates from Woods.—Wood distillates are the most important flotation agents; they, and especially the essential oils obtained by steam distillation, are particularly good effervescing-agents, but they are relatively very expensive. From resinous woods the lightest of the steam distillates is turpentine, which is water-white in colour and has a gravity of about 860—870, and of which the yield is about 6 gallons per ton of wood; turpentine is not much used in flotation, being expensive. Next to distil over would be pine oil, of a deep straw-colour and having a gravity of about 920—950; about 2 gallons of this oil would be recovered per ton, this small amount accounting for its relatively high price; pine oil is one of the most widely-used agents in flotation. Then, from the residue, about 150 lb. of resin per ton could be extracted by a suitable solvent, naphtha or gasoline, for instance. Of the destructive distillates, before charring of the wood began turpentine and pine oil would come over, in amounts similar to those stated above. Following these and the charring of the wood, commercial pine-tar oil, brown to black in colour, would be recovered to the extent of about 40 gallons per ton. Upon refining, this commercial oil would yield 15 lb. of wood creosote, cherry-red in colour and with a gravity of 940—990; and 25 gallons of pine-tar, brown to black in colour and heavier than water.

Eucalyptus oil has properties very similar to pine oil; it is about 7 per cent soluble under the conditions of dilution employed in flotation. That most employed, obtained from *Eucalyptus amygdalina* and *Eucalyptus dives*, contains relatively much phellandrene, which, being a turpene, oxidizes gradually to a gummy or resinous matter.

Hardwood distillates include pyroligneous acid, from which acetic acid and alcohol are obtained industrially, and an oily tar. This hardwood tar consists of about 12 per cent pyroligneous acid, 8 per cent wood oil, 40 per cent wood creosote, and 40 per cent wood pitch, these fractions being capable of separation. The gravity of the wood oil is about 935, while the wood creosote is heavier than water. In the supernatant pyroligneous acid there is also a dissolved tar which can be recovered to the extent of about 3½ gallons per ton of wood. Of the settled tar about 7 gallons per ton of wood is obtained.

Of these wood-distillates pine oil is that most widely used; it is essentially an effervescing-agent, and has little property selectively to coat a mineral in preference to gangue; the froth it forms is somewhat brittle, the bubbles are small and not often larger than half an inch. Eucalyptus is a similar oil but to some extent it also acts selectively. The amount in which these two oils are used is very small, less than one pound per ton of ore ordinarily sufficing, particularly when the contaminated water is returned to the circuit.

Wood creosote possesses both effervescent and selective properties. The fraction possessing the latter gives also greater elasticity to the froth and greater size to the froth bubble, two to three inches being a common size. Since this selective fraction departs with the concentrate, more wood creosote is necessary than pine oil.

Pine-tar oil and wood-tar oil secure effervescence, exercise selection, and give film-strength. The last two effects being the more pronounced, it is generally necessary to add an essentially effervescing agent, such as pine oil. Being thick and heavy they require prolonged mixing and at least an ordinary temperature; they are not satisfactory in cold water.

Finally, wood pitch or resin dissolved in soda-ash or other alkali, is a very effective effervescing-agent, capable, where no special selective agent is necessary, of raising a clean and complete concentrate.

Coal-tar and Derivatives.—Coal-tar may by distillation be divided into, light oils to the extent of some 2·5 per cent; middle oils, 18·5 per cent; heavy oils, 9 per cent; and pitch, 70 per cent. From the light oils, naphtha, benzene, toluene, and a residue are obtained. From the middle oils are separated: small proportions of phenol (carbolic acid), cresol (cressylic acid), and residue; naphthalene salts in greater proportion; and fuel oil to the extent of about 65 per cent. The heavy oils consist of over 80 per cent of green fuel-oil, the remainder consisting of anthracene and some residue. All the different residues mentioned are largely creosote, one ton of average tar producing about 300 lb. of this commodity.

Tars containing this amount of creosote or more may be used crude, provided they are added in the grinder and an ordinary temperature prevails; otherwise they do not become sufficiently emulsified. To counteract the thickness and viscosity of tar a solvent such as kerosene is often added. Among tar derivatives, cresol is an excellent effervescing-agent, though without much power of selection; being expensive and only produced to the extent of about one and a half gallons per ton of tar, this derivative has now been largely replaced by tar itself, the use of which is considerably cheaper even in the larger amount necessary. Coal-tar creosote is largely used; it effects both effervescence and selection, giving also a tough froth with an elastic bubble about two inches diameter. Exceptionally, such coal-tar chemicals as alpha-naphthylamine, xylidin, etc., have proved successful as effervescing-agents[1] (Figs. 333, 335).

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[1] Robie, *E. & M.J.*, Nov. 1, 1919, p. 730.

Mineral Oils and Refinery Products.—Crude petroleum may by distillation be separated into: gasolene with a gravity of 600—725; benzene, 725—745; naphtha, 745—790; kerosene, 790—820; fuel oil and paraffin; and asphalt. Acid treatment of the kerosene removes tarry matters; this acid treatment consists in agitation with a small amount of strong sulphuric acid, the acid now rendered black being drawn off after settlement. Such acid-sludge, as it is termed, sometimes contains as much as 50 per cent of free acid.

Mineral oil is never an effervescing-agent, though some crude oils with an asphalt base and a high sulphur content have given a sparse mineral-froth. Having a well-marked affinity for sulphide minerals, it is, however, a good selective agent; the heavier fractions, fuel oil or residue, being preferred because being thick they stiffen the froth. It is sometimes stated that mineral oil gives a non-selective froth; it gives this froth, however, because, not being an effervescing-agent, the selective action of a multitude of air-bubbles does not develop, gangue and mineral alike become entangled in the oily bubbles at the surface. Kerosene is sometimes used as a dilutant to tar and tar oils. Gasolene has been used to clean the ore particles of any resinous or sticky covering. Acid-sludge has found considerable use both to stiffen the froth and for the acid it contains. All these products demand the concomitant addition of an effervescing-agent.

Fixed Oils and Fatty Acids.—The fixed oils comprise fish oil, sperm oil, whale oil, cotton-seed oil, rape-seed oil, palm oil, etc.; these, while they have an affinity for sulphide mineral, are expensive and disagreeable to use. Practically the only agent of this group which has been seriously applied is oleic acid, obtainable from fats by treatment with sulphuric acid. This acid, sometimes known as spirits of soap or as candlemaker's oil, is thin and mobile at ordinary temperatures and gives very fair effervescence; it has, however, only been successful in hot solutions and with heavily mineralized ore. Stearic acid and palmitic acid, homologues of oleic acid, are solid at ordinary temperatures. Acetic acid, the lowest of this series, gives good effervescence but has not been applied commercially to flotation. Similarly, amyl alcohol, an alcohol high up in its series and known familiarly as fusel oil, though a good effervescing-agent, has not been applied in flotation.

Organic Salts.—Principal among organic salts are the soluble soaps, sodium oleate for instance. These are selective rather than effervescing, and generally require the help of a pronounced effervescing-agent. In the presence of soluble earthy or metallic salts, insoluble greasy soaps are formed such as might selectively coat oxidized ores present. As will be mentioned later this idea has been followed in the endeavour to float tin ores.

The use of organic electrolytes, such as phenylglyoxine in an alkaline solution, and sulpho-salicylate of aluminium, has recently been suggested. These electrolytes becoming ionized in solution, it is possible, by proper choice of electrolyte, to activate with the metallic or equivalent ions the mineral it is desired to float, while the radical ions act as the effervescing-agent. This pregnant suggestion has been applied with success to the flotation of tin ores, though so far only on a laboratory scale; it finds further mention later when dealing with the flotation of oxide ores (p. 452).

This group of flotation agents forms a link with that of the flotation chemicals shortly to be described.

Flotation Mixtures.—Generally speaking, froth flotation, being air flotation, depends primarily upon the effervescing-agent and only to a secondary extent upon the other agents. This agent, as has been pointed out, is a substance largely soluble in itself, or the soluble fraction of a more insoluble substance; the selective agent, on the other hand, is a substance largely insoluble, or the insoluble fraction of such a substance. A single substance, eucalyptus oil or creosote, for instance, may therefore be capable of serving both functions. Experience shows, however, that where a single substance might be effective of itself, the same effectiveness is more cheaply and consistently obtained by a mixture, compounded to provide, respectively, the necessary effervescence in the water, a selective film on the mineral, and elasticity for the floated froth. To-day, accordingly, mixtures are more common, single substances being exceptional. In such mixtures the selective and froth-strengthening agents, which pass away with the concentrate, enter more largely than the purely-effervescing agent which in greater part remains with the water, and is returned for further use; they also, as a rule, include cheap unrefined products whose soluble fractions by agitation become available for effervescence, saving in that way some of that more expensive agent. A single substance added alone would generally be of the more expensive class demanding sparing use, and, accordingly, the operation would be relatively delicate and sensitive to fluctuations of feed and agent; this delicacy the greater quantity of the mixture avoids, since the more expensive effervescing-agent is then distributed through the cheaper selective agent. Such mixtures, nowadays, are made up of about three parts of selective agent, say coal-tar or fuel oil, to one part of effervescing-agent, the exact proportion depending upon whether a strong effervescing-agent such as pine oil, or one less strong such as creosote, is used. It may of course be considered that crude coal-tar, if it contain 20 per cent or so of creosote, is a sufficient mixture in itself and of justifiable use alone; that may be so, but experience has shown that constancy of the agent is best obtained by compounding a mixture to specification. If a mixture be made up partly of a volatile oil, care should be taken that volatilization did not take place.

In the selection of a suitable substance or mixture many factors enter. Cost at once precludes such substances as turpentine, acetic acid, olive oil, etc., the cost of which might jeopardize or seriously diminish the operative profit, however effective the agent might otherwise be. The substances chosen are either those refinery by-products for which no special competitive demand exists, or unrefined products cheap by reason of their abundance. The movement from refined products to crude materials, and from single substances to mixtures, is well illustrated by the transference of favour from cresol to coal-tar creosote, then to coal-tar with pine oil, and finally to coal-tar with creosote. Fuel oil (mineral) and wood-tar oil together now make a common mixture. In the matter of cost, locality has much to say; it has determined at least that in Australia eucalyptus oil is largely used, in America pine oil, while in Korea camphor oil has been used.

External Factors influencing the Choice of Flotation Agents.—The influence of the mineral itself upon the choice of the flotation agent lies in such facts as, that the desired cleanliness of a copper concentrate is less than that necessary for lead or zinc; that galena has such an affinity for air that little selective agent is necessary; that blende is so laggardly in floating that the agents require to be intense. According to an investigation by Varley,[1] in the United States, coal-tar, kerosene acid-sludge, and pine oil are the principal agents for copper ores; hardwood creosote, coal-tar, crude petroleum, and pine oil, for lead ores; pine oil, hardwood creosote, for zinc ores; and fuel oil, pine-tar oil, and turpentine, for gold and silver ores.

The size of the material also influences the choice of agent, in that fine gangue is likely to be flocculated with the fine mineral if much selective agent be present, and in that coarse mineral requires a strengthened film to keep it suspended in the froth till safely delivered.

The type of machine also influences the choice, since where the risen froth has farther to travel to reach discharge it must be stronger; further, the violence of agitation will determine whether a thick or a thinner mixture must be used, greater violence more readily encompassing the required emulsification and spread of thick agents.

Finally, in selecting the flotation agent, the subsequent treatment both of the concentrate and of the tailing must receive consideration. If the concentrate has subsequently to be cleaned by water or to be subjected to cyanidation, a tarry agent might give rise to difficulty; if it has to be cleaned by further flotation, the too copious froth which an unadulterated effervescing-agent would give, would be a nuisance. Even where no cleaning is necessary there might be difficulty in breaking down the stiff, voluminous, and coherent froth which tar sometimes gives. Similarly, a thick oil would probably interfere with any subsequent cyanidation of flotation tailing (p. 480).

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[1] U.S. Bureau of Mines, Reports of Investigations No. 2203. Abstracted *E. & M.J.*, Feb. 12, 1921, p. 303.

Amount of Flotation Agent.—The amount of agent added should be the minimum capable of producing effective contamination of the water on the one hand, and of the mineral on the other (p. 423); more than that would not only be useless and an unnecessary expense, but harmful (p. 424). An excess of pine oil, for instance, makes the bubbles still more brittle and breaks down the froth; an excess of mineral oil causes gangue to be taken into the foam, or, if unemulsified, it agglomerates the sulphides so that they are in danger of sinking. The stickiness of oil is not a property of use in froth flotation, indeed care must be taken that it is not exercised; a greasy froth settles down and lies flat. An excess of tar might conceivably not be so harmful, since it is not so greasy.

The minimum amount will be that which under the quality of the agitation gives the necessary milkiness to the water, this milkiness sometimes amounting to a prolonged opacity. Agitation and the amount of agent are accordingly somewhat interdependent, prolonged agitation requiring less agent.

As already indicated (pp. 395, 423), the amounts used vary from a few ounces to a few pounds per ton of ore, the smaller figure applying to ores which need little or no selective agent. Exceptionally it is more than one per cent of the ore, but generally that excessive amount would be to circumvent the Minerals-separation patent granted for the use of oil up to one per cent.

The above figures of consumption pertain to a normal dilution of the ore pulp and to conditions which include the return of the tailing water to the circuit, making available the contaminant remaining in that water; where such a return is not made the consumption will be greater, particularly that of the effervescing-agent. The cost of flotation agents is generally from 1½d. to 2½d. per ton of ore treated.

FLOTATION CHEMICALS AND CIRCUITS

As already stated, flotation 'chemicals' are mostly inorganic salts and mostly electrolytes. Unlike flotation agents they are rarely used to secure effervescence. Common salt above, say, a 3 per cent concentration is capable of so doing, as witness the foaming crests at sea. Other chlorides and carbonates possess similar properties and like common salt have been used in the laboratory as effervescing-agents.[1] Flotation chemicals are however used rather as selective agents, either like normal selective agents directly to increase the non-wetting properties of the mineral to be floated, or inversely, to increase the wetting power of the water so that the gangue may more surely sink. Used in this latter service they may be described as "modifying" agents.

Since one chemical may under different conditions serve differently, it is more convenient to describe the services of the more commonly-used chemicals under their own names.

Sulphuric Acid.—Principal among these modifying agents is undoubtedly sulphuric acid, this chemical having been used from the introduction of flotation at Broken Hill. In treating the zinc middlings on that field, sulphuric acid, indeed, appeared to be more than an auxiliary agent, flotation not being effective without it; quite apart from its generation of gas-bubbles in the Potter-Delprat process it was of almost equal importance in air flotation, rendering floatable—probably selectively by electrolytic action but possibly by removing films of iron oxide—much blende which otherwise would have been left in the tailing. Improvement in recovery by the use of acid has been experienced elsewhere and with other ores, at Anaconda for instance with copper ore.[2]

It is probable, however, that the more general use of acid to-day is, as a modifying agent, to minimize any tendency of the gangue to float. Acid does this by increasing the wetting power of water, loosening the attachment of air bubbles to the gangue; at the same time it reduces the volume of froth, effecting in consequence a saving in those selective agents which leave the pulp and go with the froth. Acid is also said to deflocculate the pulp, rendering free thereby small mineral-particles which might otherwise remain entangled in floccules, to the betterment of recovery and to the greater cleanliness of the froth. Ordinarily, acid is rather a flocculator than a deflocculator—it has, for instance, been used to promote the settlement of slime—but it is conceivable that in flotation this normal effect is masked or even reversed by the presence of the other agents; observation certainly shows that when acid is used the water below the froth remains turbid and does not clear as it would in the presence of a flocculator. Finally, it is quite possible that acid may serve to correct the fouling of the solutions with soluble portions of the ore or impurities in the mill water; it might, for instance, bring about the oxidation of ferrous sulphate, a substance deleterious to flotation.[1] Acid may also be beneficial in nullifying any inopportune chemical emulsification of the oily agent.

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[1] *M. & S.P.*, Feb. 2, 1918, p. 167.

[2] Laist and Wiggin, *Trans. A.I.M.E.* Vol. LV., 1917, p. 492.

The use of acid in flotation constitutes what is known as an "acid circuit," acid being added to render the pulp acid to litmus, a condition reached when the amount of free acid is about 0·1 per cent of the water. The consumption of acid will, of course, depend largely upon the character of the ore, being larger where calcite or any similar carbonate is present; when the amount of such reacting substance is excessive, an acid circuit is no longer possible. Experience shows that the normal consumption is much the same as that of the ordinary flotation agents, namely, from 2 to 5 lb. per ton of ore, and that a consumption equal to about 1½ per cent of the weight of ore is one which few ores could bear. It is interesting to note that in the United States acid is used to a greater extent with copper ores than with lead or zinc ores, the total consumption with copper ores being such that divided over the total tonnage of such ores treated by flotation it amounts to 3 lb. per ton; also that, contrary to the practice in Australia, it is little used with zinc ores, whereas with lead ores it is moderately used.[2] Apart from its cost, the disadvantages of acid are: that it is corrosive, necessitating special liners of wood or cast iron in the mixing appliance; that, as already mentioned, an acid circuit precludes the addition of the ordinary agents to the grinding machine, which thereby is no longer available as a mixer; and, that by reason of the salts then taken into solution and deposited afterwards, the porous medium of pneumatic machines requires more frequent cleaning and renewal. Sometimes acid is harmful to recovery; when oxidized copper minerals are present, for instance, copper sulphate is formed which, if not precipitated, unfavourably affects the flotation of copper and silver sulphides. The greatest use of acid appears to be with slime, its presence with such material keeping the gangue from going into the froth. Generally, also, it may be said that an acid circuit gives better control of the operation and is more reliable than a non-acid circuit, since the amount of acid can be varied as appearances indicate. In cleaning the particle-surface acid is only useful with material which has suffered incipient oxidation, dump material and dry-crushed material, for instance.

Sulphuric acid is used in greater amount than any other chemical or agent in flotation; its place is sometimes, though not often, taken by nitre cake.

Caustic Soda.—Where for any reason acid is impossible and the circuit still requires correction, an "alkaline circuit" is employed. Foremost among the substances available for the production of such a circuit is caustic soda, which while relatively cheap is effective. This chemical benefits the operation chiefly by increasing the wetting power of the water, a cleaner concentrate resulting; it also deflocculates the pulp so that fine mineral-particles are not entrapped in floccules. It accordingly behaves much like acid. At the same time it undoubtedly has powers of cleaning mineral surfaces and of correcting foul mill-water; when, for instance, lime is present in amount sufficient to cause flocculation, caustic soda would correct this tendency; or, when copper sulphate forms and is harmful, it precipitates the copper as innocuous sulphide. This chemical finds extended use notably with the disseminated copper ores of Arizona, where the consumption is about one pound per ton, and with the complex contact ores of Korea.

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[1] *M. & S.P.*, Dec. 18, 1915, p. 931.

[2] Varley, Reports of Investigations No. 2203, U.S. Bureau of Mines. Abstracted *E. & M.J.*, Feb. 12, 1921, p. 303.

Sodium Carbonate has much the same effect as caustic soda, and in the form of soda-ash or the natural trona, finds considerable use in the United States.

Lime is sometimes used to correct the presence of heavy-metal sulphates, reacting with which it forms soluble calcium sulphide; care must be taken, however, that the alkalinity does not exceed about 0·02 lb. of calcium oxide per ton of water or there may be undesired flocculation and a voluminous but lean froth. Ordinarily consumption is about 3 lb. per ton of ore.

Sodium Sulphide.—In addition to the more common alkaline substances just mentioned, the use of sodium sulphide has at times proved beneficial. At the Belmont-Shawmut gold mine, California, for instance, while deflocculating the gangue this chemical is credited with flocculating the auriferous sulphides[1] (p. 484). It probably also has power to clean mineral surfaces, and is capable of converting lime into a soluble sulphur compound with no power of harm. The amount in which it is added is generally about 0·5 lb. per ton of ore. A still more interesting use of sodium sulphide sees the formation of a sulphide film over oxidized particles, rendering them floatable; this "sulphidizing" of oxidized particles is discussed later.

Sodium Silicate.—An alkaline circuit is also obtainable by sodium silicate, that is, water-glass, the addition of which so increases the wetting power of water that undesirable sulphides may be prevented from floating. Its use, for instance, might permit the recovery of chalcopyrite from pyrite and pyrrhotite, or the recovery of galena in preference to blende, the latter being recovered subsequently; such "differential flotation" of sulphides is discussed later. Sodium silicate is, however, so strong a wetter or deflocculator that its use may give rise to difficulty in the subsequent settlement of the concentrate.

Potassium Cyanide, so valuable in the metallurgical treatment of precious-metal ores, has not in general been found of any benefit in flotation; certainly, the attempts to float cyanide tailings have not been successful, only a poor recovery being obtained. Exceptionally, it would appear capable of correcting water fouled with copper sulphate.

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[1] Parsons, *M. & S.P.*, Nov. 6, 1920, p. 661.

Ammonia has been found to be a useful modifying agent when the flotation agent is wood-pitch dissolved in soda-ash.

Copper Sulphate.—When neither acid nor an alkaline salt is added the circuit is described as a natural or "neutral circuit." Among neutral salts copper sulphate is the only one of importance in flotation. This chemical has been found particularly beneficial in the flotation of zinc blende, probably because it deposits an incipient film of copper sulphide upon the blende, though it also appears capable of selectively flocculating the fine particles of zinc blende while leaving the gangue free.[1] At the Butte-and-Superior, Montana, and elsewhere, the addition of copper sulphate is, for instance, part of the established practice.[2] Its use was first suggested by Bradford in Australia, who found that fine blende particles were readily floated by the addition of about half a pound of copper sulphate per ton of ore, though coarse particles remained unfloated unless acid were added. This sulphate has, like acid, also been found useful in re-conditioning selective agents which inopportunely had become wholly emulsified. On the other hand, it has been found harmful in the flotation of silver and of copper ores. The amount in which it is added is very small and generally about 0·2—0·5 lb. per ton. Apart from the use of copper sulphate as described, there are possibilities of using it to effect a copper filming of those minerals in relation to which copper is strongly electro-negative; cassiterite particles in contact with metallic zinc and submerged in a solution of copper sulphate quickly become covered with a red film of metallic copper. This "metallizing" effect is further mentioned under oxide flotation (p. 452).

Flotation Circuits.—The particular circuit employed will, it is seen from the foregoing, depend upon the nature of the ore and upon the size of the material treated. At Anaconda, for instance, sand flotation is largely undertaken in a neutral circuit, slime flotation, on the other hand, in an acid circuit. At Utah Copper, again, the sand is treated in an alkaline circuit, whereas acid is used with the slime. Exceptionally, where both an acid circuit and an alkaline circuit have, for one reason or another, been impossible, a beneficial cleaning effect has been obtained by the addition of gasoline or other hydrocarbon in a neutral circuit.[3] An acid circuit would, perhaps, always be preferred if it were possible and other things were equal. Its use would be disadvantageous if cyanide treatment followed. It also appears impossible to use it when the ordinary flotation-agent is alkaline, as wood-pitch dissolved in soda-ash. The acid circuit found so essential to the recovery of blende in Australia gives place to a neutral or alkaline circuit in the United States, copper sulphate taking the place of the acid. An alkaline circuit gives a less viscous froth, more easily handled.

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[1] Ralston & Yundt, *Amer. Chem. Soc.*, Sept. 1917. Abst. *E. & M.J.*, Oct. 27, 1917, p. 751.

[2] Callow, *Trans. A.I.M.E.* Vol. LVI., 1917, p. 687.

[3] Atckison, *M. & S.P.*, April 27, 1918, p. 576.

The acid circuit was that first employed; then the neutral circuit, the employment of which was signalled at the Kyloe mine, New South Wales, in 1910;[1] and finally the alkaline circuit, introduced about the year 1913.

It will be realized that flotation chemicals generally are electrolytes, and that though they are at times beneficial by reason of their chemical action, they are probably of greater benefit by their electrolytic action. Seeing that this latter action has not the variety of chemical action, it is not surprising that acid and alkali have often produced similar effects.

The addition of chemicals to correct mill water, generally necessary even when only fresh water flows through the circuit, is decidedly more necessary when, as so often happens, the water which has completed the circuit is returned for use again. Without such necessary correction deleterious elements would accumulate to the detriment both of recovery and enrichment.[2] A point deserving mention is that, in process of grinding, fine metallic iron finds its way into the circuit, particularly when steel balls are used in preference to flint pebbles. In some instances, and particularly when treating slime, this fine iron has been found to be beneficial, a benefit arising, it is supposed, from its neutralizing effect upon acidity inopportunely present.[3] Elsewhere such fine iron has been found to be of no benefit, or exceptionally of harm.[4]

Finally, temperature may be regarded as a modifying agent. Flotation can be practised in an ordinary or in a "hot circuit." In ice-cold regions the circuit is generally maintained at an ordinary living temperature, since a low temperature would necessitate prolonged mixing or might even render thick agents impossible; low temperature, greatly increasing the viscosity of the water, would also cause such an amount of gangue to be retained in the froth that the resulting concentrate would be markedly unclean. But, though the advantages of an ordinary temperature above an ice-cold temperature may justify the expense of heating, the use of hot circuits has not been continued. In a hot circuit the end-point of the operation is more quickly reached and is more definite; viscosity of the water is decreased and the concentrate is cleaner; the agent is more mobile and goes farther. In the early days of flotation at Broken Hill the hot circuit was favoured because oleic acid did not work well in the cold; to-day, however, the benefits which the hot circuit would bring are more economically obtained by a proper choice of agent, and where necessary by thinning the agent with a mobile solvent.

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[1] Ashcroft, *Trans. I.M.M.* Vol. XXII., 1913, p. 1.

[2] *M. & S.P.*, Dec. 18, 1915, p. 931.

[3] Gahl, *Trans. A.I.M.E.* Vol. LV., 1917, p. 605.

[4] Coghill, *M. & S.P.*, Feb. 9, 1918, p. 197.

DIFFERENTIAL FLOTATION

Differential flotation takes advantage of the different floatabilities of minerals, floating one mineral ahead of or in preference to another, and, if desired, floating a second mineral subsequently. The floating of one mineral in preference to another may be described as the "Preferential Flotation" of that mineral, but the whole subject, to include, for instance, the recovery of two minerals in succession, is best described as "Differential Flotation." This differential flotation works on no different principle from ordinary or "Collective Flotation," but, sulphide having to be separated from sulphide instead of from ordinary gangue, the differences in floatability are finer, the necessary agents more choice, the operation more delicate, and the results less precise.

Differential flotation may be applied either to an undressed ore-pulp or to a collective concentrate obtained from a previous flotation. Treating the latter, it is not applied ordinarily to material coarse enough to be separated by gravity concentration, but only where, by reason of mineral fineness or lack of sufficient difference in gravity, water separation is not possible. When achieved by so varying the agents and electrolytes that the more floatable mineral floats while the less floatable mineral sinks, it may be described as normal; such normal differential flotation is generally applied to ore pulps. When, however, advantage is taken of other properties, the tendency to oxidation, for instance, the more floatable mineral may be made to sink, while the less floatable mineral floats; such irregular flotation, making use of more costly means, can only be applied to collective concentrate.

Normal Differential Flotation.—The idea of floating sulphides successively in the order of their floatability was first suggested and put into operation by Lyster in 1913 at Broken Hill, to treat slime, the overflow from classifiers. In this material the galena particles were smaller than the blende particles, both because such would be their natural relation in the classifier overflow, and because the greater portion of the coarse galena had already been removed by water-concentration. The galena being smaller would for that reason alone be more readily floated, but to that favourable circumstance was added its greater natural floatability. These two factors together made it comparatively easy to set the conditions so that while the galena floated the blende sank. These conditions included a minimum of eucalyptus oil, a neutral circuit, a moderate agitation, and the control of the air supply at its entry below, the result being a relatively weak yet voluminous froth, from out of which the blende particles inopportunely entangled had ready fall; moreover, the froth area at the overflow was constricted, enabling the galena to crowd out any competing blende. This suggestion of Lyster has since been carried further, and to-day at Broken Hill, galena is being obtained as a preferential float by cascade machines from re-ground jig middlings, this practice replacing separation previously accomplished by tables and vanners (Fig. 315).[1] With the galena removed, more intense conditions raise the blende; that is to say, more oil, greater violence of agitation, and the acid found necessary at Broken Hill, readily float the blende

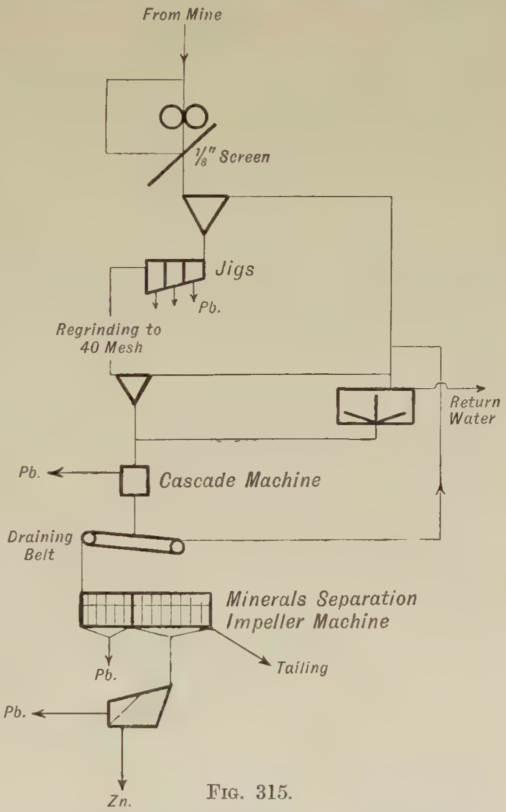

FIG. 315.

Differential Flotation at the Central Mine, Broken Hill.—Flow-sheet, 1917. Referring to Fig. 225, illustrating the Collective Flotation previously practised, it will be noticed that tables and vanners have been displaced by the introduction of Preferential Flotation of the lead. A comparison of the results obtained under the two policies is as follows:

	Recoveries.	
	Collective Flotation.	Differential Flotation.
Lead .	77·6 per cent.	85·6 per cent.
Zinc .	85·8 ,,	83·3 ,,
Silver .	49·2 ,,	64·4 ,,

In these figures neither the lead in the zinc concentrate nor the zinc in the lead concentrate is counted as recovered (pp. 377, 421, 439). (Harvey, *Trans. I.M.M.*, November 1918.)

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[1] Harvey, *Trans. I.M.M.*, Nov. 1918.

together with any small amount of remaining galena. This blende flotation is undertaken in Minerals-separation machines, the first cells of which produce a concentrate of the remaining galena.

The differential flotation of galena and blende has also been effected in the Coeur d'Alene district, Idaho, by using a small amount of effervescing agent to float the galena, and then in another machine a selective agent and often some copper sulphate to raise the blende. Experimentally it was also found that both common salt and sodium carbonate would act as combined effervescing-and-selective agents to float the galena preferentially.[1] In this connection it is interesting to note that Freeman of Broken Hill suggested the preferential flotation of galena from a lead-zinc slime by a 3 per cent solution of sodium carbonate, blende being afterwards raised by the addition of small amounts of oil and copper sulphate.[2]

Galena has also been separated from blende by manipulating the froth overflow from pneumatic flotation machines. With the Callow machine, for instance, the froth is so tender and its depth so great that, of a mixed galena-blende froth building up at the bottom, only the galena might arrive at the top. If, then, a skimmer were set to take off the top two or three inches, the blende, not being taken, would of necessity fall back, eventually to be recovered in a separate cell. In such a scheme of differential flotation two circuits would be maintained and kept separate, a lead and a zinc circuit, each being fed with its proper flotation agents and served with its own water (Fig. 316).[3]

Differential flotation is again instanced in the separation of chalcopyrite from pyrite and pyrrhotite.[4] These minerals, when occurring together, are commonly so intergrown that, being so much of the same gravity, water-separation is impotent except for rough work, beneficiation hitherto having been by hand-picking or jigging, to obtain a roughly-enriched product. Tests made upon an ore containing about 3 per cent of cupriferous pyrite and 11 per cent of pyrrhotite, in a schistose gangue, showed that 5 lb. of sodium silicate per ton without the assistance of any oil was sufficient to sink the pyrrhotite while raising the cupriferous pyrite; sodium and magnesium chlorides in solution, and even sea-water, were similarly successful. These electrolytes were not, however, capable of floating the bornite of another ore, nor the chalcocite of a disseminated ore, nor, finally, the chalcopyrite of a quartzose ore.

With other ores, tests have shown that an ordinary frothing oil may be capable of raising chalcopyrite as a preferential float from pyrrhotite and pyrite.[1] An insoluble oil may, however, not be used, or all the sulphides will agglomerate and preferential flotation be rendered impossible.

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[1] Zeigler, *E. & M.J.*, April 20, 1918, p. 741.

[2] W. Shellshear, *M. & S.P.*, Oct. 27, 1917, p. 616.

[3] Rice, *E. & M.J.*, Sept. 14, 1918, p. 482.

[4] *M. & S.P.*, Feb. 2, 1918, p. 167.

Other instances of differential flotation are of molybdenite from pyrite, and of stibnite from pyrite, neither being difficult, the floatabilities of molybdenite and stibnite being pronounced. It may, however, be remarked that molybdenite is sometimes separated from associated pyrite by skin or film flotation without the necessity of froth as a collector; to make this separation precise, the crushed material is dried and exposed

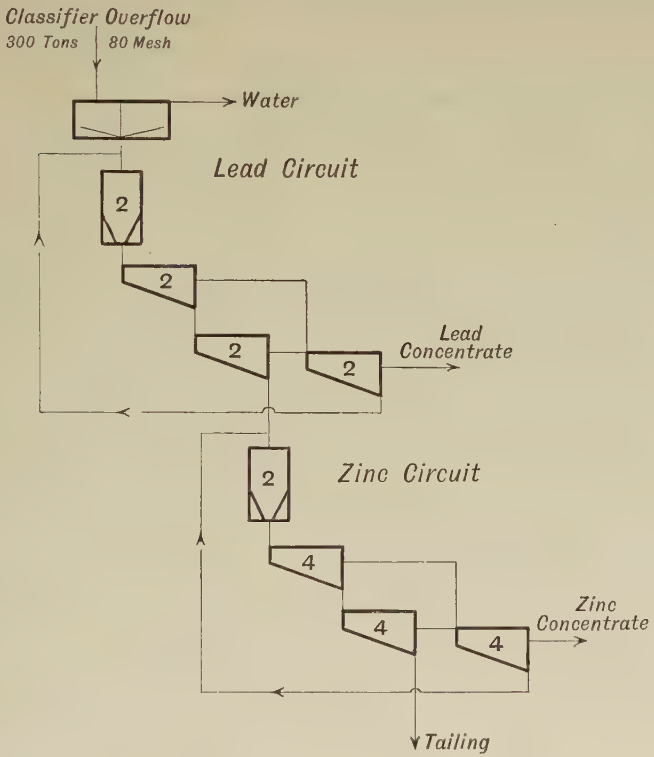

Fig. 316.

Differential Flotation at Morning, Idaho.—Flow-sheet. The air used in the pneumatic agitators (Brown or Pachucha tanks) was at 15 lb. pressure; that used in the Callow machines was supplied by a Roots blower at 5 lb. pressure. An additional air-lift was required in the pneumatic agitators to keep the sand from settling. Acid was used to keep down the siderite; Cleveland-Cliffs oil was used for the lead, and pine oil for the zinc. The numbers in circles represent the number of machines for the stated daily tonnage. The Callow machines were 2 by 10 feet (p. 440). (Rice, *E. & M.J.*, September 14, 1918, p. 482.)

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[1] Coghill, *M. & S.P.*, Feb. 9, 1918, p. 195.

to air, the pyrite then becoming covered with an incipient oxidized film so that it readily sinks.

When the two minerals to be separated are both very fine, as with impalpable slime for instance, and differential flotation receives no help from difference in size, some reinforcement to wetting may be necessary to encompass the separate sinking of the less floatable mineral. With such material Bradford at Broken Hill discovered that sulphur dioxide so deadens the sulphides in succession that, by controlling the entry of this soluble gas, the blende could be sunk while galena still floated. This idea was put into operation at Broken Hill in September 1916 to treat 500 tons of slime per day.[1]

In the Bradford sulphur-dioxide process the slime to be treated is made into a pulp consisting of 3 of water to 1 of dry ore, to which pulp sodium hyposulphite is added to the extent of 1·5 to 2 lb. per ton of ore, and sufficient acid to acidify. This mixture passes to a mechanical agitator to give time for the generation of the sulphur dioxide, and the contact of this gas with the blende. Leaving this mixer the acidity should not be more than 0·03 per cent, nor the temperature more than 90° F. Thus prepared, the mixture is sucked into an agitator resembling a centrifugal pump, air and further sulphur-dioxide from special sulphur-burners being drawn in through a snifting valve. In the centrifugal agitator the air becomes churned into the pulp, the aerated mass being then delivered through the side into a tall flotation box, with a constricted frothing area above and a controlled discharge below. Here galena overflows as a thick coherent froth, while the tailing discharges into the next agitator and flotation box, retreatment being continued till the end point of the lead separation is reached, that is to say, till the froth which in the first box had been 2 or 3 feet deep, has become so shallow that without water it can no longer overflow. About nine such treatments are required for the lead, all the froth proceeding as a single product to be settled and filter-pressed, the liquors being returned to the head of the circuit. Towards the end a little more acid is added to maintain the acidity, and the temperature is again brought to its maximum of 90° F.

The tailing, now containing only the blende, passes to a thickener where supernatant liquor is separated for return to the lead circuit, and a thick underflow is discharged. This latter passes into vats where it is kept from settling by revolving paddles, and raised to a temperature of about 125° F. by steam coils. It passes thence to a mechanical mixer where, with returned water from the zinc circuit, it is brought to a dilution of 4 of liquid to 1 of solid, and sulphuric acid is added to bring the acidity to 0·3 per cent, reckoned on the liquid; all the little remaining gas here escapes.

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[1] Henderson, *M. & S.P.*, Sept. 28, 1918, p. 407.

Sometimes, when all the blende is very fine, about 0·2 lb. of copper sulphate per ton of material is added instead of acid, the circuit then being neutral. After mixing, the prepared pulp is drawn away by centrifugal pumps which, sucking in air at the same time, deliver their aerated product into flotation boxes of the same type as in the lead circuit. Of such centrifugal agitators and flotation boxes there are six in series, the froth diminishing from great depth in the first to an indefinite film in the last. From this last box the final tailing is discharged, to be settled and thickened, the liquor being returned. The overflowing zinc froth, unlike that of the lead, is not a finished product, but requires cleaning. This cleaning is accomplished in another series where the dilution is greater, 12 to 1, and the acidity less, about 0·1 per cent; that is to say, where the conditions are such that at Broken Hill entrapped gangue is dropped.

REPRESENTATIVE RESULTS FROM THE BRADFORD SO_2 PROCESS

	Zinc. Per cent.	Lead. Per cent.	Silver. Oz.
Feed	16·5	12·8 *	15·5
Lead Concentrate	8·0	63·0	84·0
Zinc Concentrate	50·0	4·0	13·0
Tailing	2·0	8·0 †	4·2

* In addition, 4·7 per cent of oxidized lead.
† Of this, 7 per cent is oxidized lead, leaving only 1 per cent sulphide.

It is interesting to note that, as with the methods already described, the silver accompanies the lead, both metals thereby securing a better market. On the other hand, any pyrite present also goes with the galena rather than with the blende, to its detriment.

Exactly how the sulphur dioxide acts cannot yet be said. Being both reducing and soluble, it has been suggested that it diffuses into and displaces the air film by which the sulphide particles are possibly contaminated, this displacement in the case of blende being such that the tendency of the particles to attach themselves to air-bubbles is temporarily destroyed; certainly, zinc froth has been known to be killed by sulphur fumes accidentally in the atmosphere around the cells.

The preferential separation of one sulphide from another depends upon very small balances of forces and affinities. As already stated, it was, for instance, discovered by Freeman that galena could be floated preferentially in the presence of blende by the use of sodium carbonate, and without the addition of other agent or chemical, provided that the cell in which the flotation took place was either of wood or iron, as it normally would be. If, however, the operation were conducted in a copper cell or in a cell of material strongly electro-negative to zinc, or if a soluble copper salt were added, the conditions would be so altered that the blende would float in preference to the galena. In this happening there appears to be the suggestion of incipient electrolysis.

Inverted Differential Flotation.—Differential flotation is also possible by so altering the surface of the more floatable sulphide that it, and not the less floatable sulphide, will sink. Such a reversal of normal precedence may be brought about either by fractional roasting or fractional hydrochemical action.

Fractional roasting is the basis of the Horwood process put into commercial application at Broken Hill early in 1909, which makes use of the fact that some sulphides are more readily oxidized by roasting than others, and notably, galena more readily than blende. This process was applied to lead-zinc concentrate floated from slime, this concentrate being so fine that separation of the two sulphides by water was ineffective. Roasted at a low temperature, and rabbled in the presence of abundant air, the great bulk of the lead, superficially at least, is converted into sulphate, the blende remaining practically unaltered. The temperature at which roasting begins is about 350° C., after which the combustion of the sulphur carries it to about 450°. If, however, considerable pyrite be present, these temperatures may be somewhat exceeded and 500° be reached with advantage, this higher temperature being desirable to quicken the oxidation of the pyrite, in the continued presence of which the "sulphatizing" of the galena is delayed; the presence of the pyrite minimises the chances of blende oxidation. The extent to which roasting is carried varies considerably; it is only required that the galena surface should be sulphatized, but with very small particles it is almost impossible to stop at anything less than complete sulphatization. Actually, from 10 to 75 per cent of the galena is sulphatized.

The roasted mass after adequate cooling is treated in flotation cells to raise the blende, the sulphatized galena sinking, subsequently to be water-concentrated for its greater cleanliness if such be necessary. Unavoidably some small fraction of the zinc, becoming oxidized, goes into solution and is lost. Another disadvantage of the process is that to an undesirable extent the silver goes with the zinc, an association unfavourable for marketing. This undesirable happening was, however, to a large extent, corrected by giving the slime a previous water-wash.[1]

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[1] Proc., *Aus. Inst. M.E.*, No. 12, 1913, abst. *E. & M.J.*, June 13, 1914, p. 1208.

Representative Results from the Horwood Process

	Zinc. Per cent.	Lead. Per cent.	Silver. Oz.
Feed	39·0	16	18
Zinc Concentrate	49·5	6	11
Lead Residue	9·0	47	45

Fractional roasting to destroy the floatability of one sulphide in order to float another separately, was also tried at the Afterthought copper-zinc mine, Shasta County, California, upon an ore consisting of chalcopyrite and blende in a gangue containing so much barite that water separation was impossible. Differential tests made to ascertain whether the chalcopyrite, naturally more floatable than the blende, could be floated preferentially, having given discouraging results, it was decided to float a collective concentrate, and to submit this concentrate to fractional roasting, with subsequent flotation of the zinc and sinking of the copper. The scheme, however, was not so simple; the chalcopyrite had to be roasted to oxide rather than to sulphate, since not only would any sulphate formed go into solution and be lost, but its corrosive effect would be seriously felt by any iron fittings in the equipment. It is true that a small amount of copper sulphate would substantially aid in the subsequent flotation of the zinc, but getting into the main circuit it would detrimentally affect the recovery of the copper in the collective concentrate. Another disturbing factor was the solubility of copper oxide in sulphuric acid, this solubility precluding the use of that acid in the blende flotation. The whole attempt did not disclose a satisfactory process for the beneficiation of the particular ore.[1]

Fractional roasting and subsequent flotation of the less floatable of two sulphides has also been practised at the Progress mine, Leadville, Colorado, upon a blende-pyrite middling from a table treatment the other products of which were galena and tailing. This zinc-iron middling being roasted the pyrite becomes oxidized, so that upon subsequent flotation it sinks while the unaltered blende floats. Of the precious metals in this ore it is interesting to note that the greater portion of the gold remains with the pyrite while the silver is recovered with the galena.[2]

At the Trail smelter, British Columbia, a Horwood plant was erected to treat a massive galena-blende-pyrite ore from East Kootenay, but did not long remain in operation.

In general, these fractional roasting processes depend upon very careful roasting and cooling, in which the temperature range and the rate of feed are all-important factors.

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[1] Heller, *M. & S.P.*, Aug. 2, 1919, p. 151.

[2] *M. & S.P.*, Dec. 23, 1916, p. 920.

Concerning hydro-chemical differential processes, Mickle[1] showed that ferric chloride, in solution and under conditions which left the blende unaffected, was capable of so altering galena that subsequently it would sink while the blende floated. This alteration took place in the cold, requiring only agitation; the galena became covered by a very stable film of lead chloride, some sulphur, and some ferric oxide, this last arising from impurities. Though the ferrous chloride to which the ferric chloride becomes reduced could readily be regenerated, this idea has never been commercially applied.

Bradford, in his acid-salt process, proposed to attack the galena by an acidulated 7—10 per cent salt solution at a temperature of about 140° F., when sufficient sulphuretted hydrogen is evolved, let it be said, to displace the air film on the galena, whereafter, upon agitation, the blende floats. With acid in excess sulphuretted hydrogen would be generated more quickly than it could be taken up by the water, with result that the galena likewise would float. Normally, after removal of the blende, a small amount of copper sulphate is added to raise the galena.

It has also been proposed to deaden galena in the presence of blende by digesting at a temperature of about 140° F. with a 0·2—0·5 per cent solution of potassium dichromate, a chemical which will also deaden pyrite to make it separable from copper pyrite; but neither of these two ideas has been adopted commercially. Potassium permanganate, on the other hand, deadens blende but not galena, making possible a normal differential separation of the galena first and blende afterwards. Seeing that after such treatment the blende rises immediately when a small amount of acid is added, it would appear that at most only an incipient chemical alteration of the surface takes place.

The Terry Process[2] is based upon the power of ammonia in an aerated solution to promote the oxidation of metallic sulphides immersed in that solution. As the oxides of zinc, copper, and lead are soluble in ammonia hydroxide and in most ammonia salts, the floatability of these sulphides after immersion remains unimpaired, but iron oxide being insoluble, pyrite and pyrrhotite are rendered unfloatable by the film which covers them. Ammonia to the extent of about 0·04—0·06 per cent NH_3 in solution is sufficient; it may either be introduced directly or be generated in the pulp by reaction between alkalies and ammonia salts. It would generally be necessary to conduct this fractional oxidation as a preliminary treatment.

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[1] K. A. Mickle, *Proc. Roy. Soc. Victoria*, Vol. XXIV., 1911, abst. *E. & M.J.*, July 13, 1912.

[2] *M. & S.P.*, Oct. 19, 1918, p. 533.

The cost of chemicals militates against the commercial adoption of these chemical differential-processes.

Oxidized-ore Flotation

Under this heading are included the flotation of secondary oxidized ore, such as the carbonate and silicate of copper, the carbonate of lead, etc.; and also that of primary oxide ore, cassiterite, wolframite, etc.

Oxidized Ores.—Treating ore from the disseminated-copper deposits of Arizona and Utah by flotation, in addition to the recoveries made from the sulphide minerals, some 20 per cent or so of the oxidized copper-minerals, the carbonate and silicate, has usually been recovered. This recovery, while low, was made without special endeavour, the small amount of such minerals not warranting any such endeavour. Where, however, they are present in greater amount, the question has been, whether and without great expense an adequate recovery of these oxidized minerals could be made by flotation. In tackling this question the line taken has generally been to return these minerals to something like their original sulphide condition, by covering them with a film of artificial sulphide. Schwarz in America was the first to disclose this idea of "sulphidizing."

The most powerful sulphidizing agent undoubtedly is sulphuretted hydrogen. Tests made in the laboratory showed that this gas when introduced into a dry mixture containing oxidized copper-minerals, attacked them so energetically that the action could hardly be stopped till sulphidizing, not only of the surface but of the interior, was complete and the particle black to the centre; this, of course, was more than was desired.[1] Other tests showed that introducing this gas into an aqueous ore pulp containing these minerals, similar penetration of the particle did not occur, the particle becoming coated with a sulphide film, which even when most thin was yet sufficient to render the particles readily floatable. Following appropriate tests a plant was erected at the Magma Copper, Arizona, to commercially apply this use of sulphuretted hydrogen. Generation of the gas was effected by heating sulphur with crude Californian petroleum in a retort to about 300° C., the amount of sulphur thus consumed being about 3 lb. per ton of ore treated; this method was chosen after that of attacking iron matte with sulphuric acid had been shown to be more expensive. The gas so produced entered the flotation circuit at the suction of the centrifugal pump lifting the ore pulp. Contact with the minerals being thus secured, the excess gas was given opportunity to escape before the actual flotation cell was reached, experience having shown that free sulphuretted hydrogen was detrimental to the working of that cell. The results obtained from fully oxidized ore were found to be quite satisfactory, both in respect to recovery and to enrichment; it was, moreover, found that any sulphide particles still present were floated equally with the oxidized particles. But sulphuretted hydrogen is both unpleasant and dangerous, even when careful provision is made to lead away the gas escaping from a blowing cell at the head of the machine.

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[1] Callow, *Trans. A.I.M.E.*, Vol. LVI., 1917, p. 678.

The more general endeavour therefore was to exploit the possibilities of soluble alkaline sulphides. The beneficial effect of sodium sulphide in correcting the presence of deleterious substances in the ore pulp had already been experienced. Sodium sulphide, moreover, was readily obtainable commercially at a purity of about 60 per cent. Introduced into an ore pulp immediately previous to flotation it undoubtedly endows oxidized minerals with some power to float. This was convincingly shown at the Inspiration by the green froth of apparently unaltered oxidized-mineral, which formed after the introduction of sodium sulphide at a point in a multi-cell machine where the flotation of the concomitant sulphide mineral was complete.[1] It was found that this green concentrate was carbonate rather than silicate, and that for a similar flotation of copper silicate the previous addition of acid to dissolve the surface of the silicate was necessary before sulphidizing could be accomplished. An interesting fact in this experience was that it seemed immaterial whether the sodium sulphide were added before or after the oil.

Various other alkaline sulphides have been tried, and particularly sodium polysulphide obtained by treating sulphur with hot caustic soda, or by treating sodium sulphide solution with powdered sulphur; but at the Inspiration it was decided that the extra recovery of copper did not warrant the special endeavour, more particularly as the agents and conditions favourable to the oxidized minerals acted somewhat detrimentally to the recovery of the sulphides. At the Magma Copper, on the other hand, after experimenting with the soluble sulphides they returned to the use of sulphuretted hydrogen, which, it was their experience, acted favourably rather than unfavourably upon the sulphide recovery. They also found that it was immaterial whether the gas was introduced before or after the oil, an experience parallel to that at the Inspiration.

At the present moment the only mill where oxidized-ore flotation is practised on any scale is at the Shattuck-Arizona, near Bisbee,[2] where the ore consists of carbonate of lead containing chloride of silver in a gangue of quartz, specularite, and limonite. At this mill the coarse material is treated by water concentration and the slimes by flotation, the addition of about 3 lb. of commercial sodium sulphide assisting in making a 90 per cent recovery of the lead and silver (Fig. 317). The sodium sulphide is added first and the oils afterwards, no acid being used.

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[1] Gahl, *Trans. A.I.M.E.*, Vol. LV., 1917, p. 624.

[2] Allen, *E. & M.J.*, Oct. 16, 1920, p. 761.

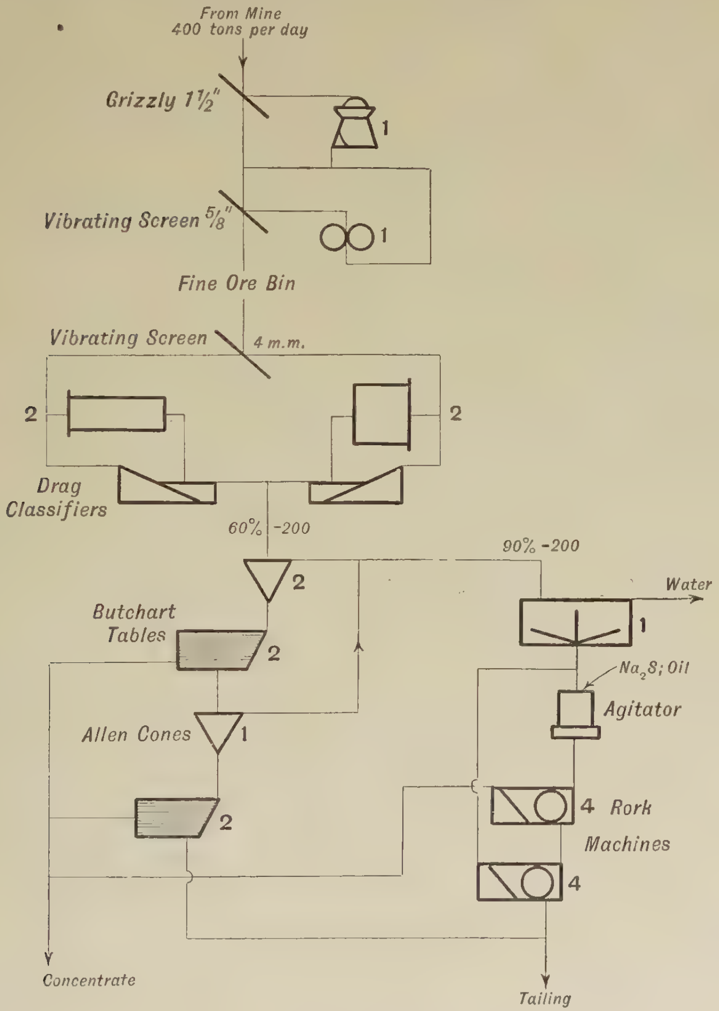

FIG. 317.

Oxidized-Ore Flotation at the Shattuck-Arizona, Bisbee, Arizona.—Flowsheet. The ore contains carbonate of lead and chloride of silver in a gangue consisting of quartz, specularite, and limonite. The average feed assays, silver, 7 oz.; lead, 6 per cent; the concentrate assays, silver, 28 oz.; lead, 33 per cent. The recovery of the silver is 71 per cent and that of the lead 94 per cent. The block figures represent the numbers of machines for the given daily tonnage (p. 448). (Allen, *E. & M.J.*, October 16, 1920, p. 761.)

Experimenting upon oxidized lead and oxidized zinc ores for the U.S. Bureau of Mines, Ralston and Allen found [1] that sodium sulphide is better than sulphuretted hydrogen for lead carbonate, though they agree with Callow that for copper, in the laboratory at least, the gas is better than the soluble sulphide; in addition to the normal sulphide Na_2S they also tried the polysulphides Na_2S_4 and Na_2S_5, and the sulph-hydrate NaSH. Their experience was that the alkaline sulphide should first be mixed with the ore in a thick pulp, that is a pulp consisting of equal weights of water and dry ore, and that only after thorough mixing should the dilution be brought up to that required in the flotation cell.

The carbonate ores of zinc do not yield readily to any similar sulphidizing. It may be that these ores contain a good deal of the silicate, and that this, like the copper silicate, is more difficult to float.

A somewhat similar process of flotation to that now being described is Leaching Flotation (p. 480). Applied to oxidized copper ores this process consists in adding first of all a leaching agent, an acid for instance, and then afterwards a precipitating agent to precipitate the copper brought into solution. This precipitating agent may be a soluble sulphide, or it may be iron shot, or finely divided iron. Flotation enters the process to recover the precipitate, that is, the copper sulphide in the one case or the cement copper in the other.[2] Ordinary metallurgical leaching is associated with iron precipitation or electrolytic precipitation. The two processes differ in that with flotation-leaching precipitation takes place in the pulp, whereas in ordinary leaching it takes place in the separated liquor.

Murex Process.—Under this name is a process, suggested by Lockwood in 1908, wherein advantage is taken of the property of the magnetic oxide of iron, magnetite, to make a paste with oil. The paste or paint so made, when used in place of ordinary oil, selectively coats any sulphides present, giving them magnetic properties by which they can afterwards be withdrawn from an ore pulp, as this pulp is passed through the field of an electromagnet. By this process not only have sulphides been recovered, but, by the addition of sodium silicate and oleine to the ore pulp, such oxidized ores as the carbonates and oxides of copper. At the Whim Well copper mine, North-west Australia, where the ore, consisting largely of sulphides, contains also a considerable amount of oxidized mineral and much aluminous gangue, normal flotation was ineffective, as also was water concentration. In these circumstances the Murex process was tried. Magnetite was made into a paint with about half its weight of oil, this paint being added at the rate of about 100 lb. per ton of ore to a thick ore-pulp, where the liquid and solid were in equal proportion ; to this pulp some oleine and sodium silicate also were added. After an adequate mixing, the pulp was taken in a thin stream upon a Zimmer conveyor to pass under an electro-magnet around which a cross-belt was running in a vertical plane at right angles to the pulp stream (Fig. 318). Coming into this magnetic field, the magnetite-coated particles and agglomerates flew upward towards the magnet, to which, however, they were prevented from attaching themselves by the intervening cross-belt. By that cross-belt they were carried out of the magnetic field to be detached with the aid of a water-spray, and dropped into a suitable receptacle. Tests made upon this ore showed that from a feed assaying 5 per cent of copper, a concentrate assaying 20 per cent might be expected and a recovery of about 80 per cent.[1]

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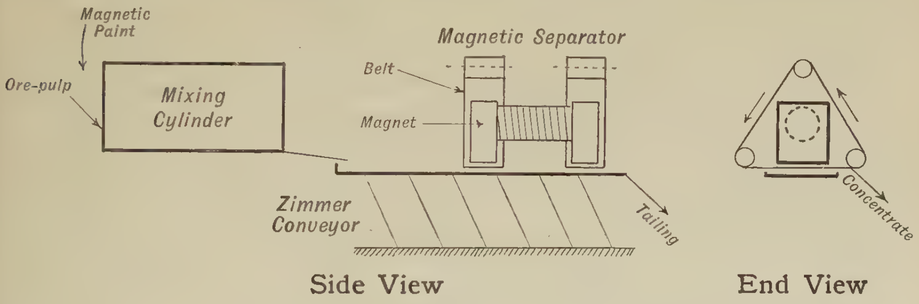
[1] 'The Flotation of Oxidized Ores,' O. C. Ralston and Glen L. Allen, abst. *M. & S.P.*, July 29, 1916, p. 171.

[2] Gahl., *E. & M.J.*, April 20, 1918, p. 717.



Side View End View

Fig. 318.

Murex Process.—Diagram (p. 450).

Better recorded is, perhaps, the application of this process at the Wohlfahrt lead mine, near Clausthal, in the Black Forest, to the treatment of fine material, water concentration being impossible by reason of abundant barite. Here again a thick ore-pulp, consisting of water and ore in equal proportion, was passed, with a small amount of a compounded mixture of oil, pitch, resin, and finely-ground magnetite, into a mixing cylinder containing small steel balls and flint pebbles. Within this cylinder the sulphide particles, becoming coated with the activating magnetic-paint, assembled themselves into floccules, which in the subsequent passage under a magnet were withdrawn in the manner described above. The results recorded show that from a feed assaying 7 per cent of lead, a concentrate assaying 60 per cent and a tailing assaying 1·2 per cent were obtained.[2]

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[1] *M. Mag.*, July 1913, p. 17.

[2] Hyde, *M. & S.P.*, June 6, 1914, p. 931.

At the Cordoba copper mine, Spain, the Murex process was adopted for the treatment of re-ground jig-middlings not amenable to finer water-concentration by reason of slime, nor to ordinary flotation by reason of calcite. With copper ores the magnetite employed may remain with the concentrate to serve subsequently as a flux. With other ores, after careful burning of the oil, it can be withdrawn by a second passage under a magnet.

Oxide Ores.—Attempts have been made to apply flotation to such oxide ores as cassiterite, wolframite, etc., minerals which ordinarily betray no affinity for air, behaving rather as stony gangue.

Among these endeavours that at East Pool in Cornwall to float cassiterite deserves particular mention. At that mine the agent employed to endow the cassiterite with powers of attachment to air-bubbles was a sulphonated fatty acid—prepared by treating a fatty acid or a soap with strong sulphuric acid,—this substance being either dissolved in such liquid as pyridine or amyl-acetate, or added solid to the flotation machine to be emulsified there with water.[1] Coated with this agent, cassiterite and wolframite are capable of attaching themselves to air-bubbles obtained by any suitable effervescing-agent. The froth formed contained in addition such other minerals, pyrite and mispickel, as normally have affinity for air. No results showing the effectiveness of the operation have been published, but it is understood that from the slime portion of the ore, assaying about 1 per cent of tin, a concentrate assaying about 20 per cent was obtained at a recovery of about 85 per cent.

The Minerals-separation experts have achieved similar encouragement in commercial tests, apparently by adding sodium oleate or a soap solution together with a curdling agent, ammoniacal liquor, for instance, the resultant curdle being selectively adsorbed by the oxide-ore particles, giving them power of attachment to air-bubbles.

Independently, A. C. Vivian, London, by using such organic reagents as cupferron and phenylglyoxine in an alkaline circuit, was able to float cassiterite. The reaction between these reagents and the cassiterite, or other oxidized mineral, is such that the metal of the mineral forms an insoluble organic salt, which remains upon the mineral surface, adsorption being undisturbed since nothing is disengaged; moreover this precipitated or adsorbed film quickly oxidizes to a tarry or gummy substance.

Cassiterite may also be rendered floatable by reducing its surface to the metallic state, a reduction which may be accomplished by heating the mineral in a reducing atmosphere, producer-gas or illuminating gas, for instance; but this idea has not been commercially attempted. As already mentioned (p. 436), it may also be "metallized" and thereby rendered floatable, by immersion in contact with metallic zinc in a dilute solution of copper sulphate, when a coating of metallic copper is deposited upon it; but, again, the idea, due to Vivian, has not been commercially applied, the copper film too readily breaking away from the cassiterite.

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[1] *M. Mag.*, April 1921, p. 252.

In this connection electrolytes have also been suggested, sulphosalicylate of aluminium, for instance. It seems possible that any salt, the metal of which would be precipitated by the mineral in question, would constitute a suitable selective agent; nor would it be necessary for the mineral to be covered with a visible film of the precipitated metal, the result apparently being reached while deposition was yet incipient, possibly by a re-arrangement of the electrostatic charges (pp. 430, 507).

Handling of Froth Concentrate

The froth which overflows is bubbly, sticky, and voluminous, sometimes relatively brittle as with pneumatic machines, sometimes tough as with mechanical machines, but never freely-flowing. Water to effect its movement down the launders is played in jets upon it, breaking it down (Fig. 305); sometimes to assist disintegration a little of the oil employed in the treatment is added, this oil in due course finding itself returned to the head of the circuit. When the pulp has to be raised to be de-watered, disintegration is also promoted by commotion in the elevators; or by treatment on tables when still-remaining gangue has to be separated, or one sulphide from another (Fig. 319). The resultant pulp generally contains about 7—10 per cent of solids, most of which. as a rule, is finer than 200 mesh.

The bubble system being broken and the mineral-particles having regained their freedom, it becomes necessary to separate the water and to obtain the concentrate in a suitably dry condition for transport and smelting. Smelters penalise concentrate which contains much moisture, a maximum of 10 per cent of moisture being satisfactory from their point of view. With fine concentrate less moisture would involve loss by dust during transport unless the concentrate were bagged, which is expensive. The whole separation of the free water may be included under the term "de-watering" (p. 295). The material being slimy, de-watering demands either intermittent settlement and decantation, or thickening followed by filtration. Thickening, by which the bulk of the free water is removed, is generally accomplished by continuous settlement, but sometimes by intermittent settlement. Any further removal of moisture is accomplished by draining and drying.

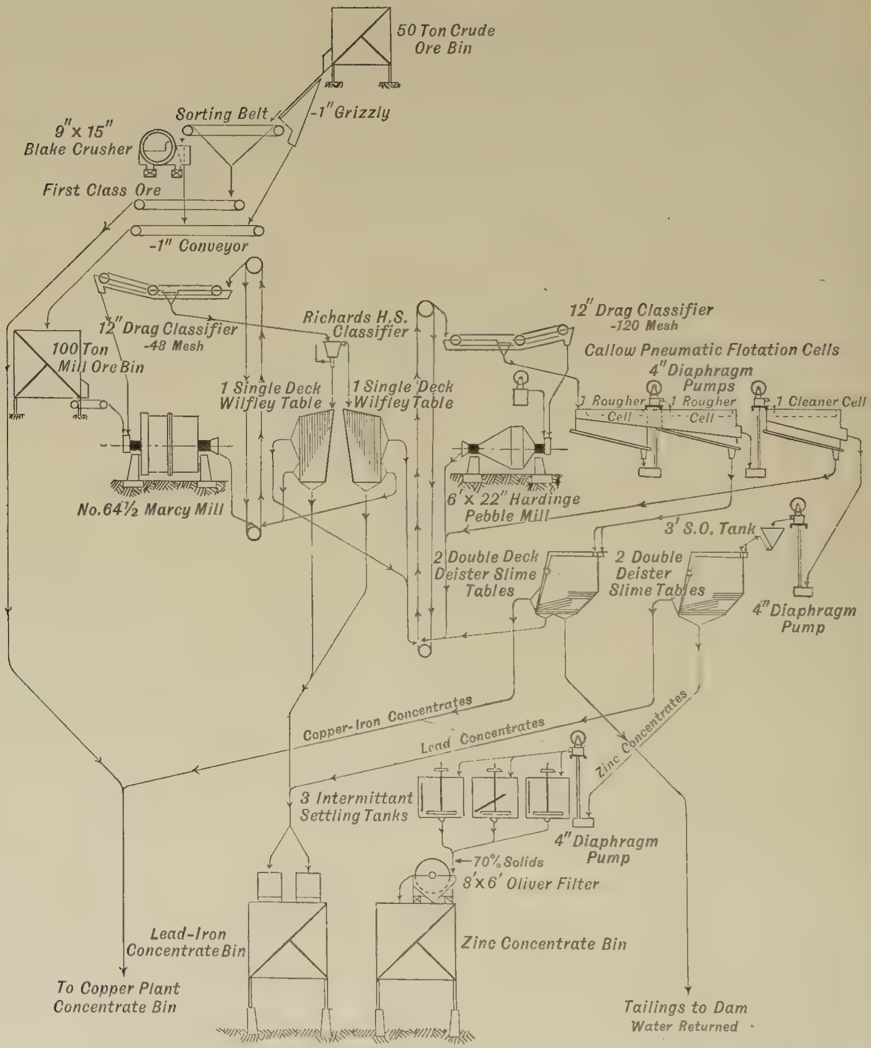

FIG. 319.

Flotation at the Magma Copper Co.—Flow-sheet of the Zinc Plant. Ball-mill crushes to 48 mesh (Tyler); conical mill to 150 mesh. Primary water-concentration recovers the coarse lead. Secondary flotation recovers the fine lead and zinc, copper sulphate improving the recovery. Final water-concentration recovers a copper-iron middling and a zinc middling sent to be reground; and discards finished tailing. In addition, the cleaned flotation-concentrate is treated on tables to separate lead from zinc, the former collecting in tanks, the latter being de-watered by settling tanks and filter (pp. 453, 455). Representative results were:

	Zn.	Pb.	Fe.
	Per cent.	Per cent.	Per cent.
Feed	12·50	8·75	3·50
Zinc Concentrate	38·50	9·25	6·50
Lead Concentrate	16·00	24·00	12·50
Tailing	3·25	4·50	0·75

(Callow, *Trans. A.I.M.E.*, Vol. LVI., 1917, p. 686.)

SETTLEMENT

Intermittent Settlement and Decantation.—Intermittent de-watering may be accomplished in rectangular tanks or in pits, of which, while one is filling and another being emptied, a third is standing quiet to give the supernatant water chance to clear and the settled concentrate time to consolidate (p. 278). Such tanks are simple appliances, the top water is easily run-off by lowering the slats of a discharge-gate ; the settled concentrate is also of desirable thick consistency, permitting digging. On the other hand, the settling area required is two or three times that which would be sufficient with continuous settlement ; moreover, the upper portion of the settled concentrate is often too thin to be handled, and requires to be separately lifted for further settlement in another tank. Sometimes complete de-watering is not attempted in a single tank but in two tanks in series, the coarse material setting in the first, the finer passing on to the second ; such a settlement in series has the advantage that when, in turn and after the first tank has received its charge, the second tank becomes the first of the series, the fine material already settled in it becomes consolidated under newly-arriving coarse material. In some places, though not often, these settling tanks have filter bottoms to permit drainage, it may be, with the assistance of mechanical suction ; sometimes also the shipping bins into which the concentrate is delivered from the settling tanks are similarly provided with filter bottoms, the concentrate draining while awaiting despatch.

De-watering may also be accomplished by intermittent thickening in conjunction with subsequent filtration (Fig. 319). So employed the settling tanks are circular and relatively deep. They are worked in the already-mentioned rotation, one filling, another emptying, and a third standing ; filling continues till the overflowing water is no longer clear, when the entering pulp is diverted and the tank stands quiet, the supernatant clear water being subsequently drawn off through a syphon. Each tank is provided with a paddle-stirrer which, after settlement, can be lowered to whip the material to a uniform consistency, and, in the circulation set up, bring it to the discharge point; discharge then is through a central spigot in the tank bottom or through a gate at the periphery. Without the aid of stirrers the bottom of the tank would be conical, discharge taking place at the apex. When the filter to which this thickened concentrate proceeds is situated low enough, discharge from the settling tanks may be by gravity, but it is more common to accomplish it with the assistance of diaphragm pumps capable of lifting it at least the height of the tank.

With intermittent settlement a single settlement is usually sufficient, particularly when filtration follows and no necessity exists to carry natural settlement to completion. When the material is fairly granular and the tanks are generous in area and depth, the consistency of the deposit, after careful syphoning of the water, may well be 80 per cent solid. On the other hand, exceptionally and when the material is very slimy, a double settlement, first in one tank and then in a second tank, may be necessary to reach the condition of 55—60 per cent solid desirable for filtration.

Continuous Settlement.—Continuous settlement is accomplished by the already-described Dorr Thickeners, wherein the settled material is brought to the discharge as fast as it settles (Fig. 185). These thickeners are worked in parallel, that is to say, each making a clear overflow ; to work them in series, a second thickener taking the overflow from the first, has been tried but found unsatisfactory. The extent of surface required for the separation of clear water varies with the nature of the material and the rate of settlement. In general, and in view of the fact that the escape of fine concentrate would be a serious matter, a much greater surface area is required when settling flotation-concentrate than when settling an ore pulp ; the ploughs of the thickener also move at a slower rate than in the settlement of ore, a complete revolution in a tank 30 ft. in diameter being made in about 9 minutes when treating granular flotation concentrate and in 15 minutes when treating slime concentrate. Greater area is also required on account of the inherent tendency of the concentrate to float. Accordingly, with lead and zinc ores, 12 to 15 sq. ft. of settling area is required per ton of concentrate settled per day, and with copper ores, these being finer and more associated with alumina, 45 to 50 sq. ft. Referring such unit areas to the tonnage of ore treated, and allowing that copper ores have a lower mineral-content than lead or zinc ores, they amount in general to about 4 sq. ft. per ton of ore treated per day.

Extent of surface being the determining factor rather than depth, the capacity of a deep tank may be increased by the insertion of trays, whereby virtually one thickener is superimposed upon another (Fig. 186). Such tray thickeners worked by a single mechanism have been found successful, but the single thickener is more commonly used.

With these continuous thickeners it has been the experience, and particularly with copper concentrate, that froth builds upon the surface, sometimes to a depth of a couple of feet. To prevent this froth from overflowing with the water, a concentric baffle, dipping 2 ft. or so below the surface and extending as much above, may be introduced, while, to prevent it building higher, water sprays are arranged. With intermittent settlers there is no similar trouble, since each charge is worked-off by itself and no opportunity exists for the accumulation of froth.

A combination of intermittent and continuous settling is sometimes seen, the coarse concentrate being de-watered intermittently and the overflowing fine concentrate thickened continuously; such a differentiation is, however, only justified where the quantities are large.

The de-watering accomplished by continuous settlers is not so complete as that by intermittent settlers, the deposited material containing as a rule less than 60 per cent of solid, particularly when handling a copper concentrate; with lead and zinc concentrates that percentage may be exceeded.

Filtration

The filtration of thickened concentrate completes the de-watering. The filters employed are adaptations from types developed in the chemical industry, and immediately from types employed in the cyanidation of precious ores. In cyanidation the service they render is the displacement of pregnant solutions by barren water-washes. In accomplishing that work, both suction filters and pressure filters are employed, some working intermittently, some continuously. It is realised that continuous suction-filters have larger capacity and work cheaper, but, as by their use the displacement of the pregnant solution is not so complete, they are only employed where the value of the liquor to be recovered does not permit the expense of a complete recovery. Their good points are, however, just those which meet the necessities of de-watering, and, to-day, the continuous suction-filter has no rival in the handling of flotation concentrate. Pressure filters, the Kelly press for instance, are sometimes used where the concentrate is more than ordinarily viscous and where, in consequence, suction would neither effect an adequate de-watering nor have a sufficient capacity; but compared with suction-filters, being almost invariably intermittent in operation, they are more costly to operate.

The type, prior and par excellence, of the continuous suction-filter is the drum filter represented by the Oliver filter, though latterly a multi-disc filter, the American filter, has found some application.

Drum Filter.—The Oliver filter consists of a hollow, revolving, cylindrical drum 8 to 14 ft. in diameter and 6 to 14 ft. in width of face, with a hollow rim (Fig. 320, 321). This rim has a backing of wooden staves laid from flange to flange and supported on interior spokes ; at regular intervals upon these staves and parallel with them are small strips to support the filter cloth laid in turn upon them ; keeping this cloth in place and protecting it from the plough which eventually detaches the filtered cake, is a close winding of fine wire. For convenience in fixing the cloth and in arranging the interior pipe service, the circumference is divided into a number of sections, some twelve to twenty. The shallow hollow space beneath the cloth connects with the vacuum pump by a number of radial suction pipes, which, at the centre, turn to follow the drum shaft towards one end of the filter. There, each pipe pierces the hollow trunnion on which the drum is built, and emerges against a channelled valve-cover, which, non-rotating itself, is held against the trunnion by a nut upon a stud proceeding from and co-axial with the trunnion. Entering from the outside of that cover, the pump suction establishes connection with the circular channels into which the suction pipes open ; in the same cover but above its centre is

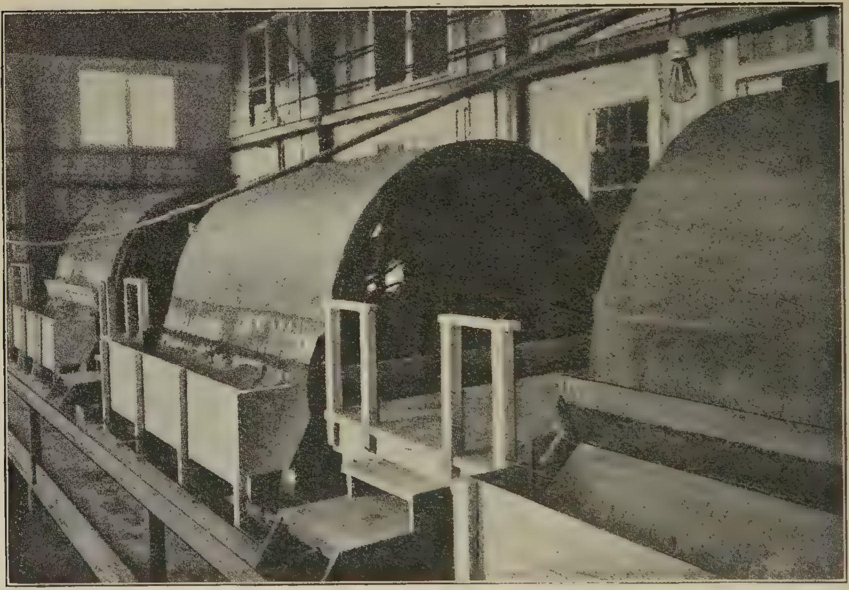

FIG. 320.

Oliver Filter.—General View. Of the three filters in the illustration the middle one is in operation, its plough engaging the drum surface. The close wiring of the right-hand filter can perhaps be seen (p. 458).

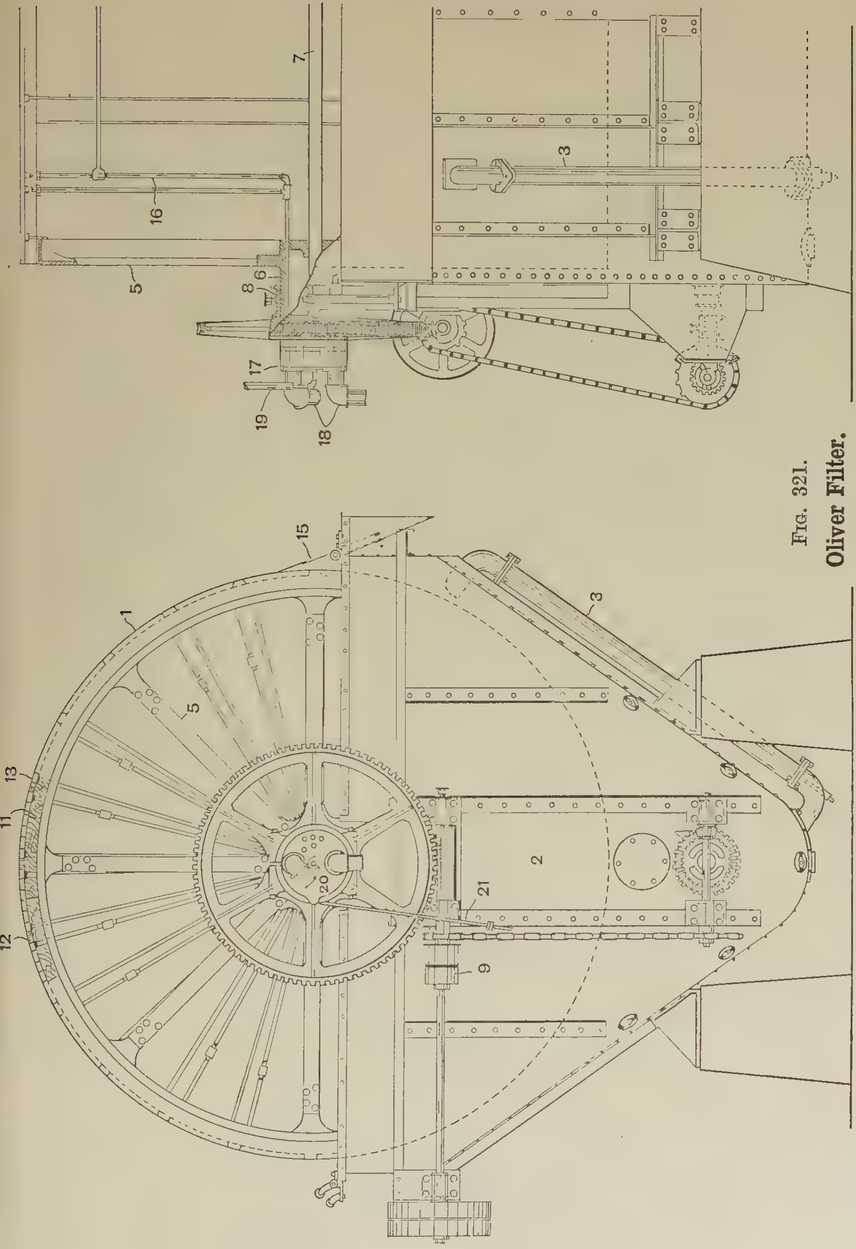

Fig. 321.

Oliver Filter.

1. Filter Drum.
2. Filter Tank.
3. Air Lift Circulators.
4. Channel Steel Drum Rim.
5. Channel Steel Drum Arms.
6. Hollow Cast-iron Trunnion.
7. Steel Drum Shaft.
8. Main Bearings.
9. Wiring Pulleys.
10. Agitator Shaft.
11. Wood Staves—Drum Shell.
12. Division Strips.
13. Filter Medium.
14. Wire Winding.
15. Steel Scraper.
16. Vacuum and Air Pipes.
17. Automatic Valve.
18. Vacuum Connection.
19. Air Connection.
20. Valve Stem.
21. Valve Adjuster.

Oliver Filter.—End and Side Elevations, partly sectional. The Oliver filter as usually constructed consists of a revolving drum or cylinder with the lower portion submerged in a tank containing the material to be filtered. The drum is supported on hollow trunnions from which radiate the arms which support the peripheral shell. This shell is hollow, being made of an inner air-tight backing of wooden staves and an outer filter cloth backed by a wire screen. The space between is divided into hollow compartments parallel to the axis of the drum. Each compartment forms virtually an independent unit, although the filter cloth is continuous. This cloth is held in place and protected from wear by a spiral wiring of wire about one-eighth of an inch in diameter, the separate coils being an inch or more apart. For winding the drum a simple device is furnished to secure the proper spacing. Arranged radially in the hollow interior are pipes connecting each compartment with the automatic valve which controls the suction and the admission of air under pressure. This valve consists of a valve plate fixed to the hollow trunnion and having holes to receive separately the pipes coming from each compartment; these holes form an outer ring. At the centre of this plate projects a stem or stud over which the valve cover is brought to be held in place by a nut and spring. This valve cover has annular ports corresponding to the 'pick-up' or suction stage and the discharge or 'blow' stage; an adjusting rod fixed to the tank prevents it from turning. Keyed to the trunnion on the valve end of the drum is a worm wheel meshing with a driving worm. The speed at which the drum is rotated varies from one revolution in five minutes to one revolution in fifteen minutes. The power required is small, being one horse-power or less for sizes less than 12 feet by 12 feet.

On the discharge end of the tank a flexible steel scraper bears lightly on the wiring to remove the cake as it is loosened by compressed air (p. 458).

a similar entry for air under pressure, the alternation of pressure and suction being accomplished automatically during revolution.

The whole drum is revolved by the engagement of a worm with a wheel fast on the trunnion, this trunnion, and a second at the other end, being supported in bearings on the sides of the pulp tank in which the drum is part-submerged.

This pulp tank usually is triangular in section, the bottom angle being rounded. It embraces the lower portion of the drum fairly closely at the sides, but more distantly at the bottom. There the pulp collects, being kept from settling by paddles upon a revolving shaft, and by an air-lift circulation. Of these two means of agitation the mechanical is the better, though without the air-lift there would be the objection, that when the filter stopped, agitation would stop also and the concentrate would be liable to pack. In a newer type the tank is semicircular, this shape giving less space in which the concentrate might pack; with this type there is an oscillating stirrer and no need for independent agitation with air. The required intensity of agitation depends upon the density of the mineral and the consistency of the pulp; a heavy concentrate requires relatively much agitation, a thick consistency lessens the necessity for much agitation. Along one side of the tank a scraper, fixed to bear upon the wire of the drum face, detaches the cake as the drum revolves, a discharge chute directing it away. The Portland filter is very similar to the Oliver (Fig. 322).

The filter cloth employed is loose twill for slime concentrate and tight duck for coarser materials. The life of a cloth is generally from three to six months, the lower figure with an acid circuit and the higher figure with a neutral or alkaline circuit. During this life it requires to be regularly cleaned with wire or fibre brushes, sometimes with the assistance of a weak hydrochloric-acid solution.

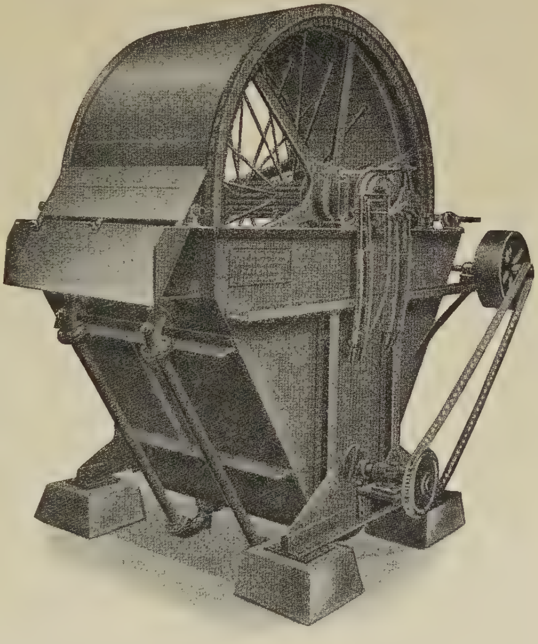

Fig. 322.

Portland Filter.—General View. The filter shown is one with a narrow face. Apparently the driving gear and the suction connection are not at the same end of the filter, the suction end being shown. The driving mechanism of the agitator in the lower portion of the tank is prominent, as also are the two side pipes of the air lift (p. 460).

The vacuum employed is generally about 24 inches at sea level and the equivalent at altitudes; it is kept high, since the degree of the vacuum determines the amount of air passing, and consequently the dryness of the cake. This vacuum is generally produced by a reciprocating pump, a dry pump being more common than a wet one; it may however be produced by a two-lobed cycloidal exhauster of the Roots type. The pump provided has generally a displacement equal to 0·5—1·0 cub. ft. per sq. ft. of filter area. The filter usually makes about one revolution in five minutes when treating sandy concentrate, and in fifteen minutes when treating slimy concentrate. Concentrate is sucked to the cloth during passage through the submerged arc of about 60 degrees, after which air is drawn through the cake till the scraper is reached, when pressure replaces suction and detachment is facilitated. Air at about 15 lb. pressure is suitable for this "blow"; steam is sometimes used.

The pulp to be filtered has generally a consistency approaching 60 per cent solid. The thickness of the resultant cake will be as little as a quarter of an inch with slimy material, and as much as one inch or more with sandy material, three-quarters of an inch being common. The capacity of the filter varies accordingly, from about 0·05 ton per square foot of filter area per day, to about 0·3 ton, being commonly about midway between these two extremes. Heat increases the capacity, and to obtain this increase the pulp in the filter tank is sometimes warmed by steam coils to 90—100° F.; unslaked lime similarly increases the capacity and is sometimes used. Neither heat nor lime, however, appears to diminish the moisture in the resultant cake, which at its best is about 10 per cent and at the worst about 25 per cent. With slimy copper-concentrate the moisture appears to increase and decrease directly with the amount of fine gangue present.

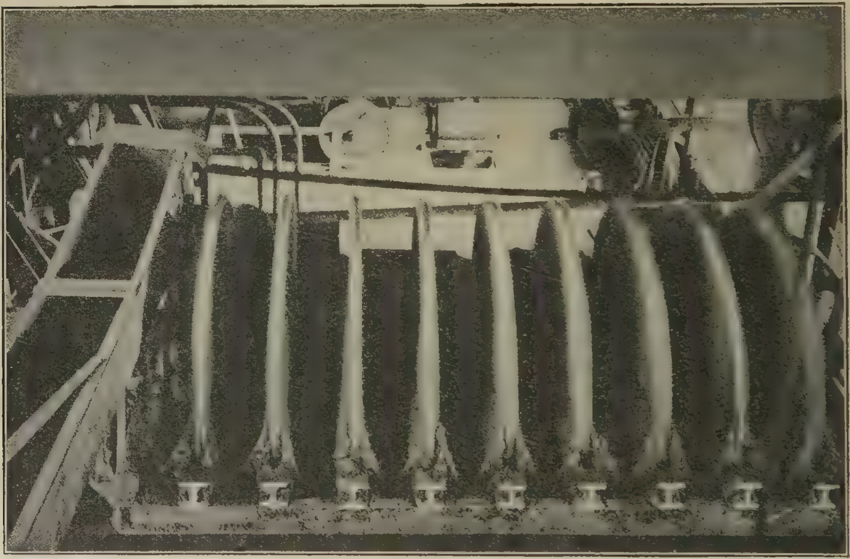

FIG. 323.

American Filter.—General View in Operation (p. 462). (*E. & M.J.*, April 20, 1918.)

Disc Filter.—The American filter consists of filter discs about 4 ft. diameter placed about 12 in. apart on a horizontal shaft (Figs. 323, 324). Each disc, divided into eight segmental filter-leaves, works half-submerged in its own pan. Each filter leaf consists of a filter cloth stretched over a pipe frame, an internal wire-screen keeping the two surfaces apart during suction. Each leaf connects through a pipe with a control valve at the end of the shaft, by which valve it is arranged that suction takes place during the greater portion of the revolution, giving place to pressure just before reaching the scraper. This filter has the advantage of large filtering area for a given floor space. Where the mill-water contains abundant depositable salts, it has the further advantage that the filter cloth can be readily removed and replaced; so light is this work of removal that it would not be unreasonable to change all the cloths every ten days. On the other hand, the discs being vertical, the cakes formed are thin at the top, where, in consequence, the air is liable to be somewhat short-circuited.

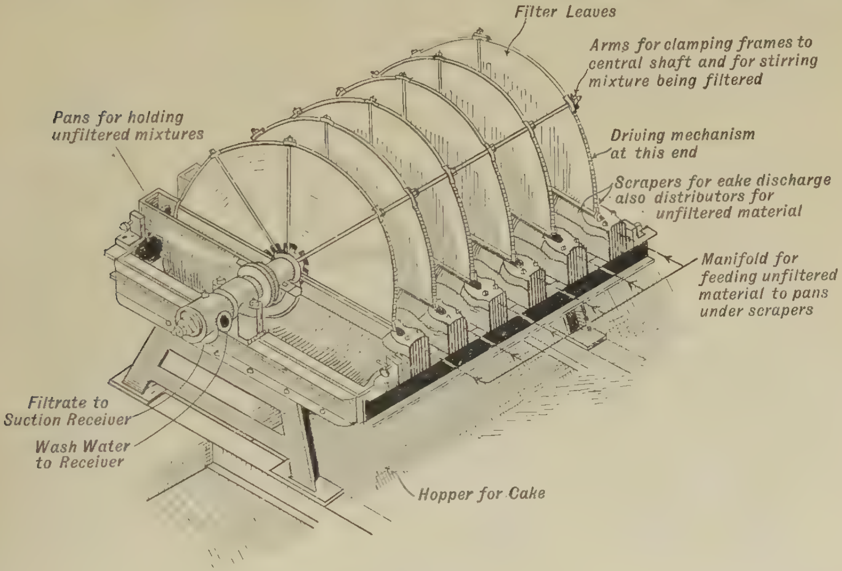
Filter Leaves
Arms for clamping frames to central shaft and for stirring mixture being filtered
Pans for holding unfiltered mixtures
Driving mechanism at this end
Scrapers for cake discharge also distributors for unfiltered material
Manifold for feeding unfiltered material to pans under scrapers
Filtrate to Suction Receiver
Wash Water to Receiver
Hopper for Cake

Fig. 324.

American Filter.—General View. Each disc or wheel is made of eight filter-leaves clamped to the central shaft by bolts extending radially from that shaft. This shaft is cored longitudinally with eight compartments. Each leaf connects with one of these compartments. At one end of the filter is the driving mechanism; at the other is a valve which connects the various compartments alternately to the vacuum and to the pressure. Shaft and discs are held in a series of cast-iron pans into which the unfiltered pulp flows, an automatic overflow regulating the pulp level. A rigid framework supports all the pans, these being brought together by clamping screws. Bosses at the pan-centres maintain a space of about four inches between adjoining pans, this space permitting the detached cake to fall.

By such an arrangement of discs great filter-area is provided within a small floor space, and large diameters are avoided. A common diameter is four feet. Any single leaf may be readily removed and covered with new cloths. The excess-solution space in each pan is reduced to a minimum, packing being thereby largely avoided. Vertical filters, however, are liable to pick up an uneven cake, thicker at the bottom, thinner at the top (p. 462).

Drying and Draining

The detached filter-cake is generally dry enough to be handled by an ordinary belt-conveyor; on large installations it is indeed often delivered by such conveyors directly into railroad cars. More often, however, the filtered concentrate is delivered into bins where some of the still remaining moisture has opportunity to drain away. While, not infrequently, it is so wet and clammy as to need drying.

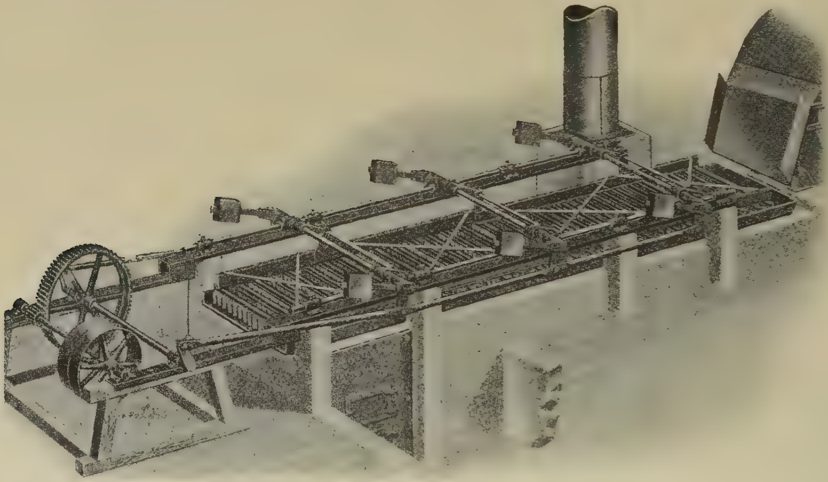

Fig. 325.

Lowden Drier.—General View. To the right is seen the drum filter in a position to discharge its cake on to the rear end of the long rectangular hearth. Suspended from above, to play upon this hearth, are transverse rakes with blades depending from them. These rakes are drawn forward and pushed backward by crank and connecting-rod, each complete movement taking place about twice a minute.

At the beginning of the forward movement the rakes are pulled down and the blades enter the material to make a cut; the remainder of the forward movement is then made with the blades in this lowered position. At the beginning of the backward movement balance-weights lift the rakes, the movement being completed with the blades in the air. The amplitude of movement being equal to about half the distance between two adjoining rakes, the blades cut into a fresh portion of the material at each downward stroke. A common size of this drier has a hearth 5 ft. wide and 17 ft. long (p. 465).

Drying is sometimes accomplished by steam coils in the bin, but this method is unsatisfactory on account of pipe troubles; moreover, a leakage of steam would be detrimental to drying. It may also be accomplished on floors with flues beneath,[1] but such a method being costly in handling is only warranted where complete dryness is required, as, for instance, when the concentrate must be bagged for shipment.

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[1] Weigall and Roberts, *M. & S.P.*, Dec. 6, 1919, p. 184.

The Lowden drier is a mechanical drier specially designed to handle viscous material. It consists of cast-iron plates with sides turned up, these plates being placed in line to form a shallow trough over a horizontal flue (Fig. 325). At one end is a fire-box and at the other end, a little to one side, a chimney-stack. On to the relatively cool end the wet concentrate is delivered, and by means of mechanically-operated rakes regularly pushed forward to be discharged at the fire-box end. These rakes move downward when they begin their forward motion, and upward to begin their return. As they move downward they cut into the drying mass, a new cut being made at each descent. The movement is quiet and no dust is created. The amount of moisture evaporated per unit of fuel burnt is generally somewhat less than that which would be evaporated by the use of such fuel under a boiler. Receiving filter cake containing 20 per cent of moisture, reasonably dry concentrate containing 8—10 per cent of moisture is produced at a moderate cost for fuel, power, and labour.

At the Bunker Hill, Idaho, a new type of drier is being tried, which consists of an iron pipe, 12 in. in diameter and 10 ft. long, set in a horizontal position for rotating. This pipe is lagged with a thickness of about $\frac{3}{8}$ in. of asbestos. Over this asbestos is wound 205 turns of copper wire weighing 90 lb. The ends of this wire terminate on collecting rings delivering a current of 82 ampères and 282 volts into the winding. The speed of rotation is about 7 r.p.m. Heat is generated by eddy currents or hysteresis in the pipe shell, this heat being prevented from radiating, by the asbestos. The copper wire remains fairly cool.

Inside the pipe are radial rabbles around an axial shaft, these rabbles extending almost to touch the shell. This shaft revolves about 1·5 r.p.m., the rabbles then effecting the progression of the material from the feed to the discharge. Drying is assisted by pre-heated air which, moving in the opposite direction to the ore, carries the moisture away. By the rolling motion the drying concentrate quickly forms into balls, which gradually break down to hard pellets, convenient for handling and transport.

In a 48-hour test 20 tons were dried, 1220 k.w. hour was consumed, and 2686 lb. of water evaporated, the moisture being reduced from 11·4 to 6 per cent.[1]

COST OF HANDLING

Including thickening, filtering, and loading, the total cost of handling is generally from 10d. to 20d. per ton of concentrate, the loading cost being the largest item. The power consumption for thickening and filtering is generally about one k.w. hour per ton. If loading is by conveyor, the extra consumption of power is about 3 k.w. hour per ton. Exclusive of the power required for the vacuum pump and for compressed air, a 12′ × 12′ Oliver filter requires about 1·5 horse-power; a 30′ × 10′ Dorr thickener requires about 0·5 horse-power at the slow speed proper to the settlement of flotation concentrate.

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[1] *E. & M.J.*, May 28, 1921, p. 909.

CHAPTER X

FLOTATION: GENERAL ASPECTS AND THEORY

Place and Part of Flotation in Dressing

Water-concentration is eminently satisfactory in the treatment of ore containing relatively coarse and granular mineral, and a light gangue; chemical treatment is equally satisfactory when the valuable mineral crushes to powder, and the gangue is not soluble. Between these two positions lies the field for flotation, this method of concentration being effective both in the presence of a heavy gangue and with fine mineral.

Flotation first established itself by the recovery of blende from an association with heavy gangue at Broken Hill, New South Wales, this success being complete about the year 1910; from the crushed ore on that field water-concentration had only been able to market galena and silver. Quickly following, in the year 1911, flotation was applied at the Kyloe mine, New South Wales, to improve the recovery of chalcopyrite from a felspathic ore, this improvement resulting from the ability of flotation to treat satisfactorily the fine and slimy material resulting from crushing; the coarse chalcopyrite continued to be recovered by water. It was similarly applied to the betterment of recovery at the Butte and Superior zinc mine, Montana, the advent of flotation putting fine water-concentrating appliances out of service; and, in 1912, to the improvement of recovery at the Britannia-Copper mine, British Columbia, and at the Braden-Copper mine, Chili.

Finally, in 1913–1914, at the Engels-Copper mine, California, flotation assumed the responsibility for all the concentrate recovered, and an all-flotation plant operated for the first time; this particular plant recovered chalcopyrite from an ore containing much magnetite (Fig. 326). Similarly, in January 1914, a mill was put into operation at the Inspiration, Arizona, wherein, for the first time, flotation was employed as the first stage in the concentration of a class of ore which hitherto had been treated wholly by water, this latter and older process being relegated to the second stage and a secondary importance.

The above developments had been with the Minerals-Separation impeller machine. Early in 1914 the Callow machine was introduced at the National-Copper mine, Mullan, Idaho, to retreat tailings from water-

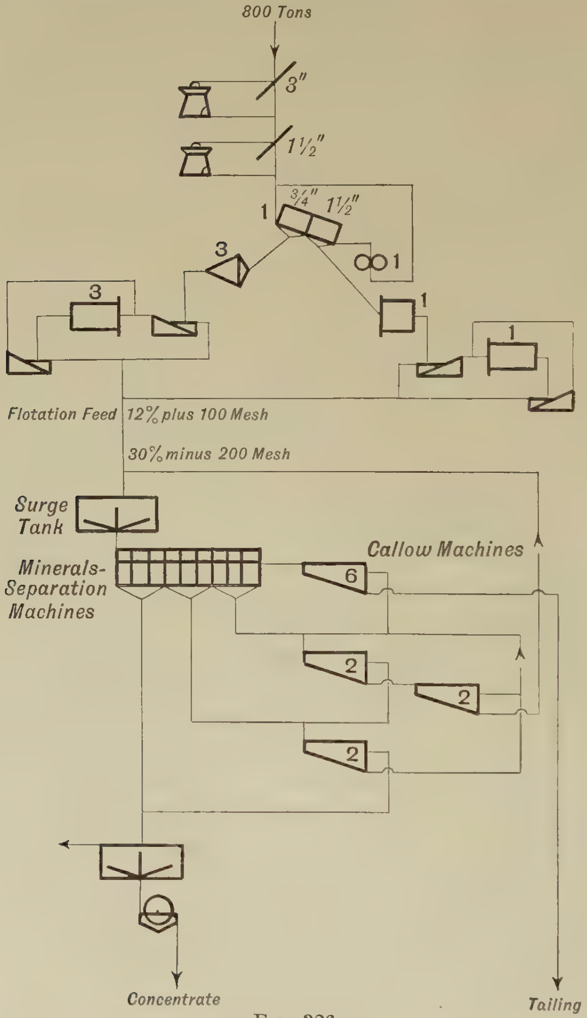

FIG. 326.

Flotation at the Engels-Copper, Plumas Co., California.—All-flotation Flow-sheet. The ore-body is a magmatic deposit of bornite and chalcopyrite in diorite, the presence of magnetite making water-concentration impossible. Results were: feed, 2·2 per cent copper; concentrate, 30 per cent; and tailing, 0·45 per cent. The block figures give the numbers of the respective machines for a daily capacity of 800 tons. This mine was the first to employ an all-flotation plant, which it did in 1914. The division of the froth of the Minerals-Separation machine into several products, and the gradual enrichment of these products, are the features of this flow-sheet (pp. 387, 467, 471, 482). (Young, *E. & M.J.*, May 28, 1921, p. 904.)

concentration, and from that date the pneumatic machine has shared in the developments. Apart from internal details, the only subsequent developments to record are the application of flotation to the treatment of primary slime from low-grade lead and zinc ores, water-concentration sufficing for the sand; and the application of cascade flotation to the retreatment of the impoverished discard from preceding treatments.

To-day finds flotation taking the following varied place in dressing:—

I. All-flotation.
II. Primary flotation; secondary water-concentration.
III. Primary water-concentration; secondary flotation.
IV. Primary water-concentration; secondary flotation; final water-concentration.

All-flotation is practised only where water-concentration is impossible, that is, where the gangue is heavy or the mineral flaky. Heavy gangue may consist of heavy silicate as at Broken Hill; of magnetite as at Engels, California; of specularite as at Swansea, Arizona[1]; of an undesirable sulphide, such as pyrite, pyrrhotite, etc., as at the Calaveras-Copper, California,[2] etc. Flaky mineral is represented by molybdenite, which though

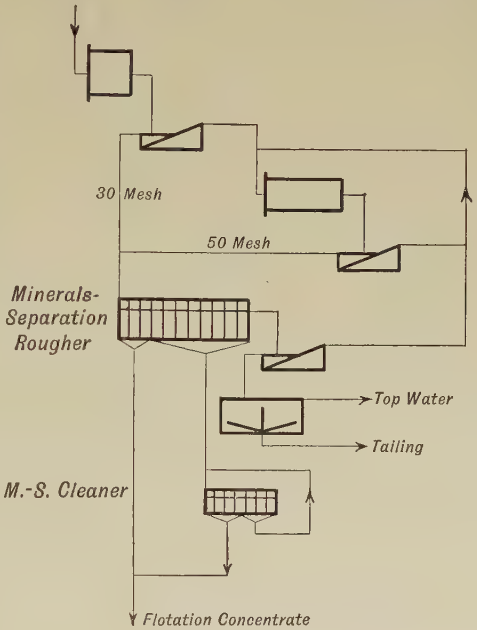

FIG. 327.

Stage Flotation of Molybdenite.—All-flotation Flow-sheet at Empire, Colorado. Ore, chiefly altered granite, assayed about 1·0 per cent MoS_2; pyrite more abundant; concentrate assayed about 25 per cent MoS_2, and tailing about 0·15 per cent. Pine oil was the flotation agent, being added to the extent of about 1·5 lb. per ton, partly at the tube-mill, partly in the first agitation-box of the flotation machine. In this instance, stage flotation consists in regrinding the coarse portion of the flotation tailing and returning the ground product to the head of the rougher machine; by adopting it and making other improvements the recovery was increased from 60 per cent to 80 per cent. The low-grade concentrate produced was cleaned in other works (p. 471). (Coghill and Bonardi, *E. & M.J.*, May 29, 1920, p. 1210.)

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[1] Thurmond, *M. & S.P.*, April 24, 1920, p. 606.
[2] Robbins, *M. & S.P.*, Nov. 25, 1916, p. 772.

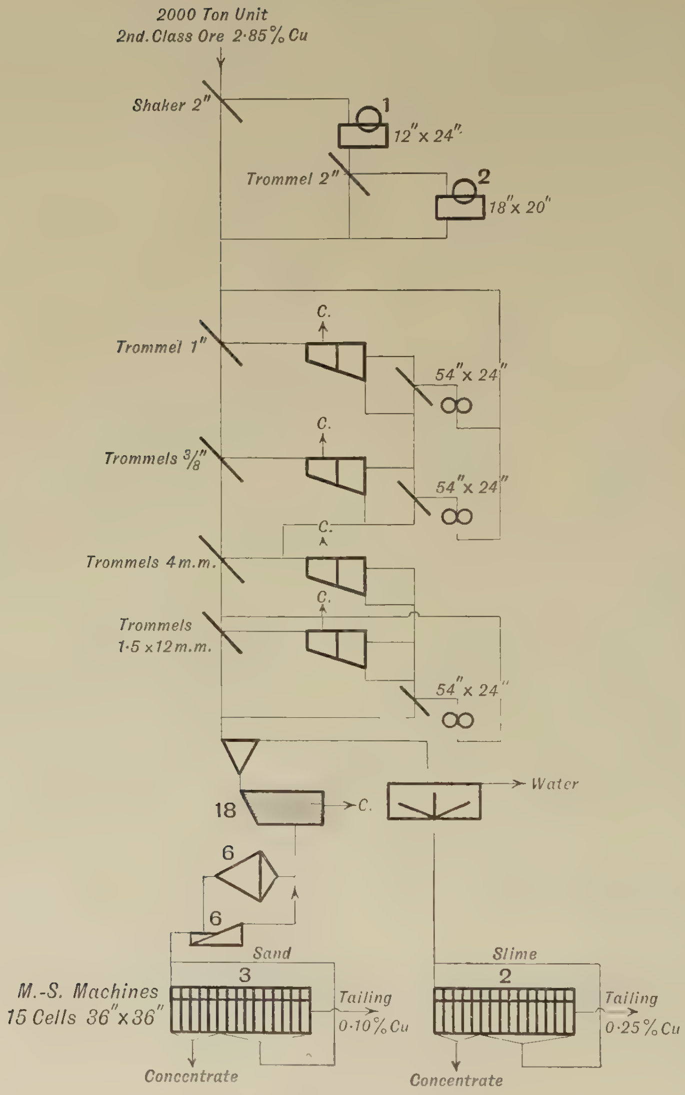

FIG. 328.

Class Flotation at Anaconda.—Flow-sheet. The sand flotation-pulp was all smaller than 2 mm., but 90 per cent remained on 50 mesh; the feed assayed 0·60 per cent of copper, and the tailing 0·10 per cent. The slime flotation-pulp assayed 2·10 per cent, the concentrate 12 per cent, and the tailing 0·25 per cent. The water-concentrate assayed about 8—10 per cent. The flotation machines, each consuming about 110 horse-power, were each capable of treating 400 tons of sand per day, or 175 tons of slime. The block figures indicate the numbers of the respective machines for 2000 tons per day (pp. 471, 475, 481). (Laist and Wiggin, *Trans. A.I.M.E.* Vol. LV., 1916, pp. 486, 646, 647.)

heavy enough sinks with difficulty and cannot be recovered by water. All-flotation may be a simple treatment of sand and slime together, such being "straight flotation" (Fig. 326); or it may be conducted in stages with further grinding between, such being "stage flotation" (Fig. 327); or conceivably it may be "class flotation," wherein sand and slime are treated separately (Fig. 328). But more of this later (p. 474).

Part-flotation schemes are, however, more common. Generally, a combination of water-concentration and flotation is advisable, the water to recover the decidedly granular mineral, flotation to recover that which is fine and approaching the impalpable. Many considerations determine the sequence of these two treatments. The natural sequence is to proceed from the coarser to the finer, that is, to allocate the first treatment to water and the second to flotation (Fig. 328). Where there is much mineral that is the usual procedure, since with much mineral some will be released at a coarse stage in the comminution; moreover, after the removal of the relatively coarse mineral enough mineral will remain to ensure a good froth in the subsequent flotation. On the other hand, with little mineral in the ore, a greater degree of comminution will generally be necessary before any adequate release is effected; so comminuted, the material will be in a condition suitable for flotation, more suitable than if some of the mineral had been withdrawn by a previous water-concentration, while any granular mineral escaping flotation may be caught subsequently by water. The two sequences are well illustrated by the respective practices at Anaconda where the enrichment effected by dressing is low (Fig. 328), and at the Inspiration where the enrichment is high (Fig. 329).

By making water-concentration the first treatment not only may coarse concentrate be recovered but worthless tailing may, if desired, be discarded; this is the practice, for instance, at the Utah-Copper,[1] Miami, etc., but more particularly with the zinc and lead ores at Joplin and S.E. Missouri, where flotation treats only the slime resulting from primary crushing, the sandy tailing from water-concentration being discarded without regrinding. Against this favourable point may be set the necessity to thicken the water-diluted pulp to the consistency proper to flotation (Fig. 330). With primary flotation, and particularly when that has included tailing retreatment, the whole flotation-tailing need not necessarily be water-concentrated but, as at the Inspiration, only the sandy portion. Secondary water-concentration has at times been regarded as a desirable corrective to irregular working of preceding flotation; such a correction is, however, not necessary where retreatment-flotation is practised; indeed, instances are recorded of secondary water-concentration being abandoned in favour of primary (Fig. 331).

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[1] Rickard, *M. & S.P.*, Dec. 7, 1918, p. 751.

The two sequences described may also themselves be combined; flotation may, for instance, come between initial and final water-concentration (Fig. 332), or water-concentration may come between two flotations. Such combinations, though more applicable to complex ores, are occasionally seen with simple ores. With an initial and a final water-concentration the former would probably be by jigs or roughing tables while the latter would

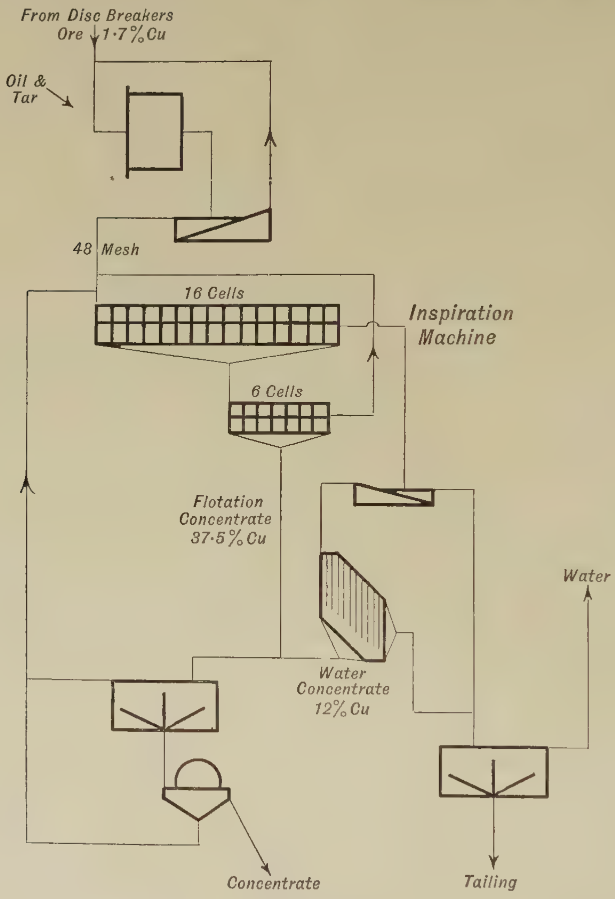

FIG. 329.

Flotation at the Inspiration-Copper.—Flow-sheet. Crushed in Marcy Mill to about 48 mesh (Tyler), primary flotation for all, secondary water-concentration for the sand. The oil mixture consists of about 95 per cent crude coal-tar and 5 per cent light wood-oil. Double-decked Deister-Machine tables recover the coarse concentrate escaping flotation. The unit mill has a capacity of 800 tons per day, there being several units (pp. 295, 377, 388, 471, 481). (Gahl, *Trans. A.I.M.E.* Vol. LV. p. 610, September 1916.)

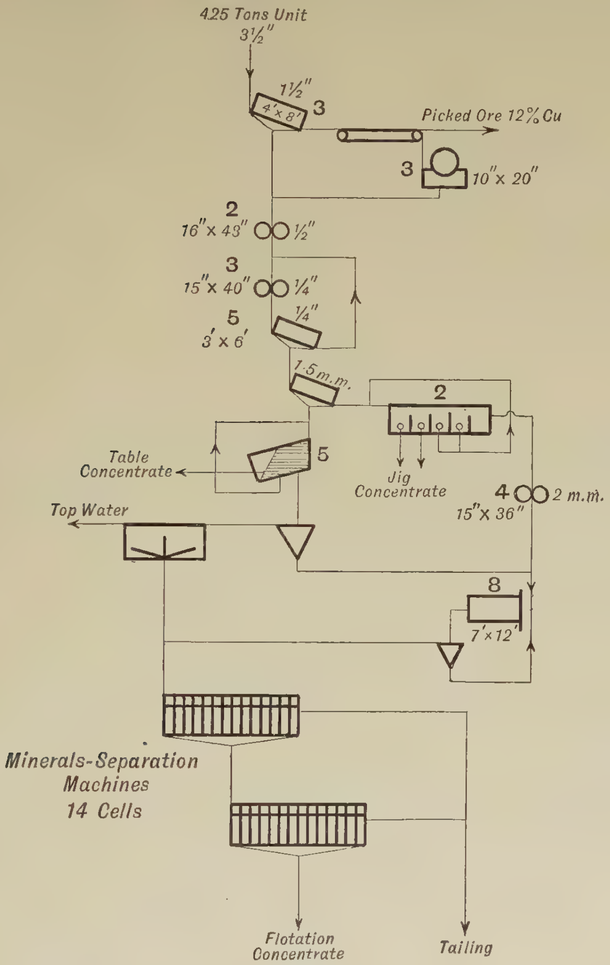

Fig. 330.

Flotation at the Britannia, British Columbia.—Flow-sheet of primary water-concentration (including jigs) and secondary flotation. Ore consists of chalcopyrite in quartz gangue, with abundant pyrite and subordinate blende and galena; country-rock is chloritic schist. The valuable products of the mill are:

Picked ore	10	per cent,	assaying	10—18	per cent	copper.
Jig Concentrate	25	,,	,,	16—17	,,	,,
Table ,,	25	,,	,,	14—15	,,	,,
Flotation Concentrate	40	,,	,,	14—15	,,	,,
	100	,,				

The flotation-feed assays 1·9 per cent copper; and the tailing 0·12 per cent. Half a pound of pine oil is used per ton, and no acid (pp. 387, 471, 473, 482, 647). (Rickard, *M. & S.P.*, November 11, 1916, p. 697.)

be on slime tables (Fig. 333). With initial and final flotation, the latter would be by cascade or other simple means (Fig. 334).

Part-flotation schemes, in addition to the advantage they present in recovering the coarse and fine mineral by processes respectively efficient for each, coarse mineral by water, fine mineral by flotation, permit also some recovery of minerals which do not readily float. Pyrite suitable for flux is thus recovered at the Inspiration; mispickel on the Suan Concession, Korea; and at places, the iron-zinc blende, marmatite.

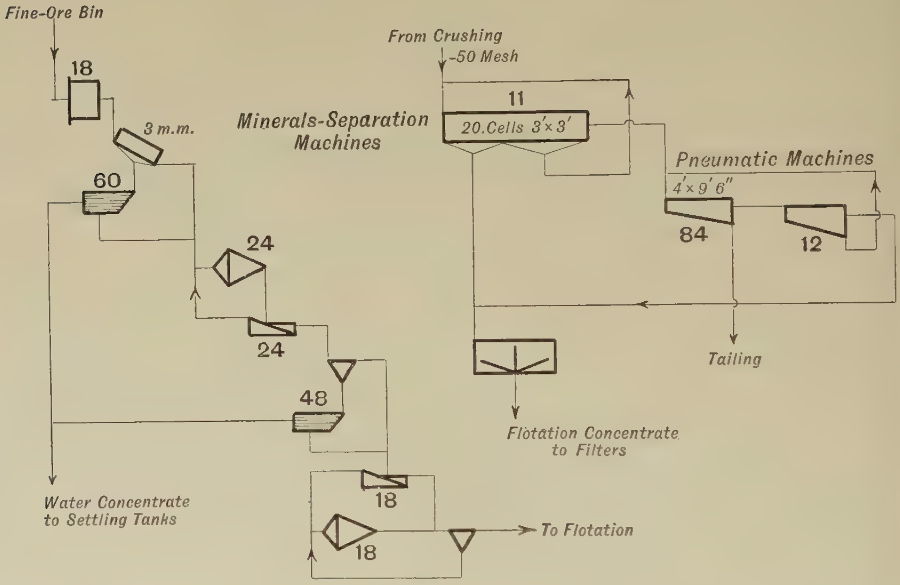

FIG. 331.

Flotation at Braden, Chili.—Flow-sheet. Primary water-concentration with secondary flotation, this procedure having replaced a reversed sequence. Ore is a shattered andesite containing chalcopyrite, some bornite, and some chalcocite. Pine-tar oil, fuel oil, and kerosene, make the flotation mixture, with acid as modifying agent. Capacity of the plant, 7000 tons per day; the block figures represent the numbers of the respective machines for this capacity (p. 471).

As with all-flotation so also with part-flotation, different procedures internal to flotation may be followed (p. 471). Stage flotation would appear justifiable where an earlier stage in comminution effects a release of mineral sufficient to warrant its removal before final comminution. Naturally, since flotation does not begin till the material has already been crushed relatively fine, say, to 50 mesh, stage flotation offers no great possibilities; the second stage is conducted after the sandy portion of the tailing from the first stage has been further crushed, say, to 60—80 mesh (Fig. 335). It does, however, tend to minimize the production of mineral of unsavable size, say, less than hypothetical 1200-mesh, and to give a coarser concentrate; yet only exceptionally has it been adopted.[1] Class flotation, that is, the separate flotation of sand and of slime, is practised exceptionally. At Anaconda (Fig. 328), employing impeller machines, it is justified by the necessity to use acid with the slime, no similar necessity existing with the sand[2]; at the Utah-Copper the practice and justification are

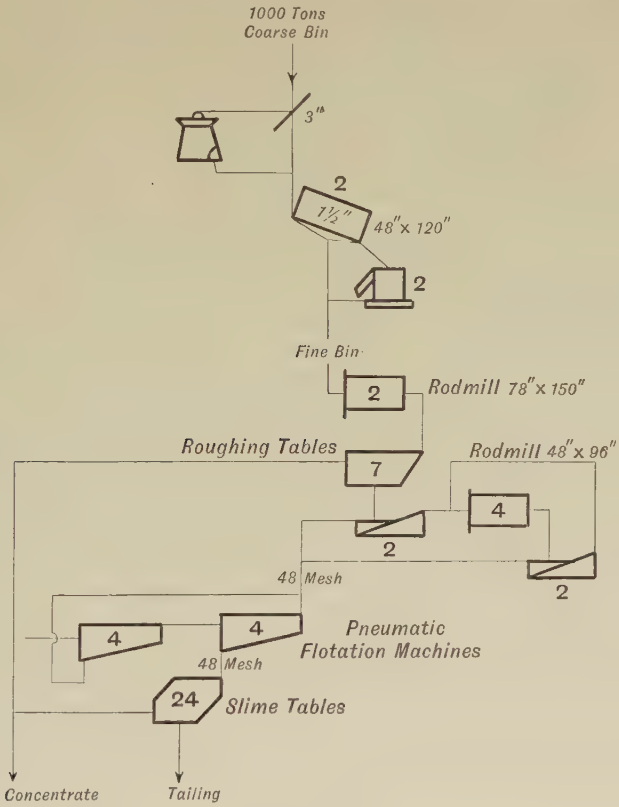

Fig. 332.

Flotation at Burro Mountain, Tyrone, New Mexico.—Flow-sheet. Primary water-concentration, secondary flotation, final water-concentration. The ore consists of quartz porphyry, quartz monzonite, etc., with disseminated chalcocite; it assays roughly 1·90 per cent of copper, of which amount one-twelfth is oxidized. Flotation is introduced between water-concentration on roughing tables and water-concentration on slime tables. The block figures indicate the numbers of the respective machines for 1000 tons per day (pp. 472, 649). (*M. & S.P.*, August 21, 1920, p. 285.)

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[1] Coghill, *M. & S.P.*, Sept. 20, 1919, p. 404.

[2] Laist and Wiggin, *Trans. A.I.M.E.* Vol. LV., 1917, p. 486.

similar.[1] Where pneumatic machines are used and much relatively-coarse mineral exists in the flotation pulp, class flotation might avoid a subsequent table treatment of the flotation tailing; even then, its adoption would only be justifiable if the tonnage were large.

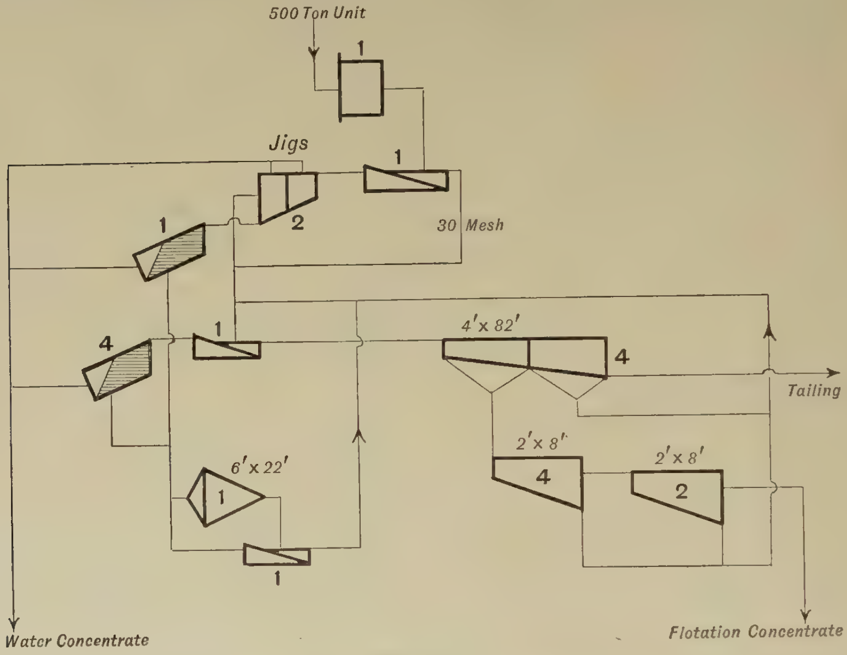

FIG. 333.

Flotation at the Arizona Hercules.—Flow-sheet. Primary water-concentration and secondary flotation. The feature of this scheme is the introduction of jigs to take out relatively-coarse sulphides, oxides, and native copper, from a disseminated copper ore, and thus to relieve the flotation plant; with X-cake as flotation agent this innovation was found beneficial. About 30 per cent of the total copper in the ore is recovered by these jigs in a concentrate assaying about 40 per cent copper (pp. 428, 474). The results obtained were:

	Sulphide Copper.	Oxide Copper.	Metal Copper.	Total Copper.
	Per cent.	Per cent.	Per cent.	Per cent.
Feed Assay	1·28	0·24	0·19	1·71
Recovery	95·0	72·5	52·0	86·0

(Shimmin, *E. & M.J.*, May 15, 1920, p. 1115.)

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[1] Rickard, *M. & S.P.*, Dec. 7, 1918, p. 751.

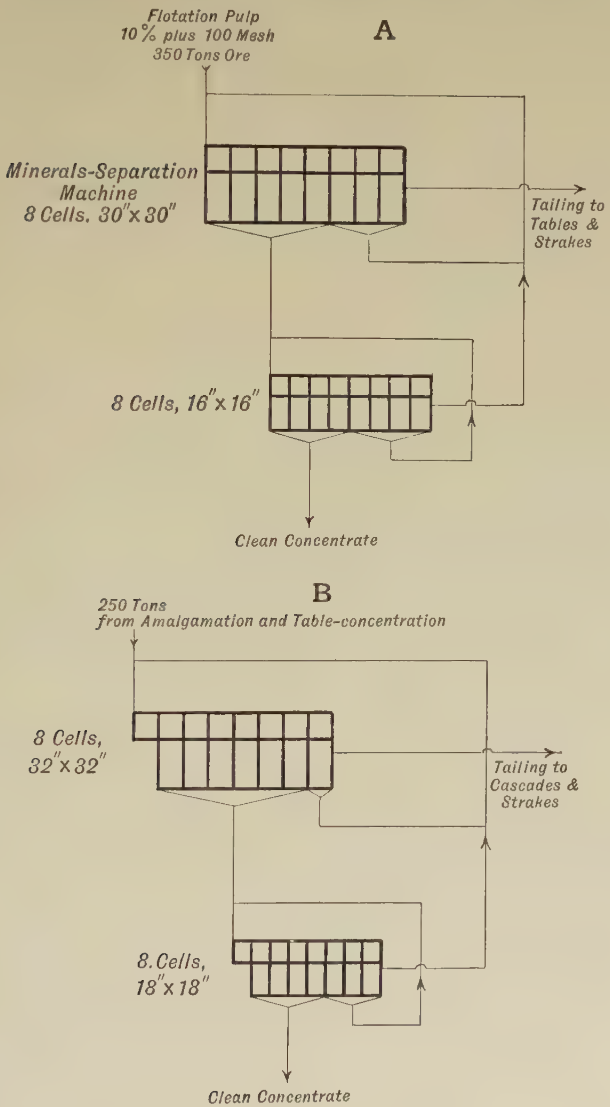

FIG. 334.

Flotation at the Suan Concession, Korea.—A, Primary Flotation at the Tul Mi Chung Mill. This mill was designed and put into operation in 1915, to make by flotation a high-grade gold-copper concentrate from a complex ore containing many of the usual contact gangue-minerals. The flotation tailing is treated on tables to recover an iron-arsenic concentrate, after which the coarse tailing is ground and passed over cement strakes. These strakes have since given place to Overstrom slime tables. In tests, no difficulty was experienced in the cyanidation of the flotation tailings. The oil used is a mixture of eucalyptus and tar; caustic soda is added. A 60 h.p. motor is provided for the rougher machine, and one of 25 h.p. for the cleaner. The flotation concentrate assays 25 per cent of copper and 3·5 oz. of gold; the table concentrate 1·5 per cent copper, 3 oz. gold, 8 per cent arsenic, and 3 oz. silver. (Weigall and Mitchell-Roberts, *M. & S.P.*, December 6, 1919, p. 812.)

B, Secondary Flotation, at the Suan Mill. In this mill amalgamation and water-concentration precede flotation. In flotation the oil-mixture is 0·25 lb. of eucalyptus and 0·20 lb. of thin tar per ton of ore; caustic lime is added to secure an alkalinity of 0·02 lb. per ton of solution. The first cell of the flotation machine is for mixing alone; it has no spitzkasten. The clean concentrate, which assays 25 dwt. of gold, 27 per cent of copper, 1·8 per cent bismuth, and 4·6 oz. silver, is tabled to separate the bismuthinite. The flotation tailing passes to cascade machines, and over canvas strakes (p. 474). (*M. & S.P.*, December 13, 1919, p. 844.)

The foregoing considerations have not included the subsidiary use of water or of flotation to further clean or divide the concentrate obtained down the main line of the flow-sheet. The water treatment of flotation concentrate, particularly when that concentrate contains two sulphides, is not uncommon; it requires that the mineral be granular. The flotation treatment of water-concentrate is recorded at various small mines where that concentrate is complex, galena having to be separated from tin and wolfram, or molybdenite from wolfram and bismuthinite; it is also recorded at the Utah Copper[1] where a vanner concentrate is thereby relieved of much associated gangue; and elsewhere.

The proportional part played by flotation varies more than its place; firstly, flotation may treat unaided all the material and be responsible for all the concentrate recovered; secondly, it may, in series with water-concentration, treat all the material yet only recover a portion of the concentrate; finally, in parallel with water-concentration it may only treat the primary slime. With its simple equipment and moderate operating-costs, flotation tends to displace water-concentration where both are efficient, even though for its proper functioning further comminution may be necessary, and even though success requires considerable manipulative skill. To any such displacement water-concentration opposes its greater reliability and the more granular nature and greater cleanliness of its products. Where the ore is coarse-grained and well-mineralized, water-concentration will not only maintain its position as the first treatment in concentration but also remain primary in respect to the amount of concentrate it recovers; where, however, the ore is fine-grained or poorly-mineralized, water-concentration, even when it retains its place in the scheme, will lose its place in importance.

The lowliest part yet taken by flotation in any scheme wherein it participates is that of treating but the primary slime. In the lead-belt of S.E. Missouri, for instance, it is considered that material down to 150 mesh is best treated by water-concentration, and accordingly only about 15 per cent of the crushed ore is submitted to flotation; latterly, however, employing ball-mills, a movement towards the greater application of flotation has been initiated (Fig. 231).[1] Similarly, on the Joplin zinc field, only the primary slime, some 25–30 per cent of the ore, is treated by flotation.

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[1] Rickard, *M. & S.P.*, Dec. 7, 1918, p. 749.

The field for flotation, as already stated, comes between that effectively covered by water-concentration on the one hand, and that given over to chemical treatment on the other. On both sides flotation has its limits; it is neither effective on material coarser than about 40 mesh nor satisfactory for material approaching colloidal size. With fresh ore there is rarely sufficient of this impalpable material to make itself felt, and flotation is entirely successful; with decomposed or clayey ore there is difficulty. An increase in the proportion of slime will quickly overload a machine.

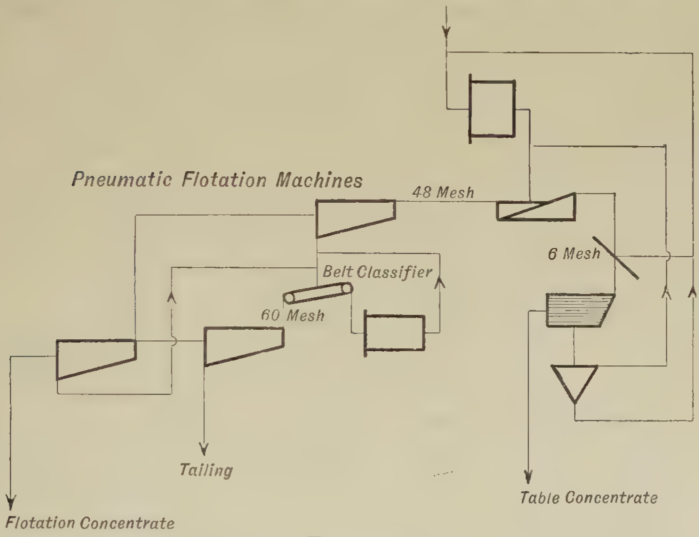

Fig. 335.

Flotation at the Arizona-Copper.—Flow-sheet to test Stage-flotation. Primary water-concentration, secondary flotation in two stages. Water-concentration of the coarse material, between 6 mesh and 48 mesh, recovered about 59 per cent of the total concentrate and 45 per cent of the total copper. The recovery by water-concentration and flotation combined was 91 per cent of the sulphide copper and 87 per cent of the total copper. Still better results were obtained by increasing the size of the flotation machines. The feature of this flow-sheet was stage-flotation, namely at 48 mesh and at 60 mesh, with grinding between. Though stage-flotation gave good results, no decision to adopt it was taken. X-cake was the flotation agent (pp. 428, 474). (Crowfoot and Witterau, *E. & M.J.*, June 19, 1920, p. 1349.)

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[1] Delano and Rabling, *Trans. A.I.M.E.*, Aug. 1920.

Impalpably fine concentrate is difficult to market.[1] Probably not much mineral finer than hypothetical 800—1200 mesh is saved or worth saving; material finer than this can generally with advantage be run to waste. It is considered better to accommodate a little oversize than to run the risk of making too much slime.

So closely juxtaposed, it is not surprising that flotation and chemical treatment overlap one another, nor that the line between dressing and metallurgy here becomes indefinite. In the beneficiation of copper ore containing both oxide and sulphide copper in substantial amount, for instance, the oxide-copper might be recovered by leaching and the sulphide copper subsequently by flotation; or, the metallic copper precipitated by fine iron from the pregnant liquor resulting from oxide leaching, might be recovered by flotation, a process which has been termed "Leaching Flotation"[2] (p. 450). Experimentally it has been shown that flotation of the sulphide copper could even proceed in the pregnant copper-sulphate liquor.[3] On the other hand, experiments at Miami suggested that though leaching after flotation was satisfactory, flotation after leaching might not be.

The combination of flotation with cyanidation has in general not proved satisfactory (p. 432). The cyanidation of flotation tailing is rarely undertaken, partly because no lower consumption of cyanide is assured, but also because the flotation agents may interfere; the tailing from a neutral or alkaline circuit is obviously in better case than one from an acid circuit. At the Nipissing mine, Cobalt, Ontario, the flotation of cyanide tailing proved unsatisfactory and was abandoned, though probably the principal unsatisfactory point was the low value of the resultant silver-concentrate. Tests elsewhere showed cyanidation to be quite effective in treating either flotation concentrate or flotation tailing, though these tests did not lead to commercial application.[4]

APPLICATION AND RESULTS

Lead and Zinc.—At Broken Hill, New South Wales, whereas before the advent of flotation the recovery was only 60 per cent of the lead, less of the silver, and none of the zinc, the recoveries to-day are about 85 per cent of the lead, 65 per cent of the silver, and 83 per cent of the zinc, these percentages not including the lead in the zinc concentrate nor the zinc in the lead concentrate. The cost of flotation is about 6s. per ton and that of complete dressing about 9s. per ton.

At the Butte and Superior, Montana, the present recovery is about 92 per cent from an ore assaying about 17 per cent of zinc, flotation being responsible for three-quarters of the production. The complete milling cost is about 7s. 6d. per ton; before the adoption of flotation it was about 5s. per ton. While flotation increased the cost per ton of ore, it increased the value of the recovery considerably more.

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[1] Pearce, *M. & S.P.*, Oct. 12, 1918, p. 491; Loth, *E. & M.J.*, Dec. 27, 1919, p. 950.

[2] Gahl, *E. & M.J.*, April 20, 1918, p. 717.

[3] Crowfoot and Donaldson, *E. & M.J.*, Sept. 4, 1920, p. 471.

[4] Rose, *Trans. A.I.M.E.* Vol. LV., 1917, p. 432.

At Joplin, Missouri, in the larger and more modern plants, flotation increases the recovery of the zinc from 62 per cent, as it was with water alone, to 70 per cent; only the primary slime amounting to about 20 per cent of the ore is at present floated; there is no regrinding.

At the St. Joseph Lead Company, of Flat River, Missouri, slime amounting to about 15 per cent of the ore, is treated by flotation. This material, 90 per cent of which is finer than 150 mesh, assays 5 per cent of lead, the resultant concentrate assays 52 per cent, and the tailing 0·7 per cent, these figures representing a recovery of about 84 per cent (Fig. 231).

At the Atlas mine, Sheffels, Colorado, treating a silver-lead ore wholly and only by flotation, the feed assays 2·5 per cent of lead and 8·5 oz. of silver, the concentrate assays 16 per cent of lead and 35 oz. of silver, while the recoveries of both are in the neighbourhood of 90 per cent, or 25 per cent better than previously obtained by water-concentration. With a capacity of about 100 tons per day the cost including royalty is about 4s. 6d. per ton.

Copper.—At the Inspiration in 1915 concentration in the mill cost 1s. 2d. per ton, royalty included; breaking at the mine cost 1·5d. per ton, and haulage to the mill 1d. The feed assayed 1·7 per cent of copper, the flotation concentrate 37·5 per cent, and the table concentrate 13 per cent, the recovery being 80 per cent. According to Gahl,[1] the cost of flotation proper was about 3d. per ton; of flotation, including table-treatment and settlement, about 10d. per ton; and of dressing in its entirety but excluding royalty, about 20d. per ton. At this mine a water-concentration plant which was under construction and promised a recovery of 70 per cent in a concentrate assaying 20 per cent, was abandoned in favour of a flotation plant which promised a recovery of 85 per cent in a concentrate assaying 25 per cent, at a working-cost 20 per cent lower and a plant-cost 25 per cent lower (Fig. 329).

At Anaconda, where in 1915 the recovery by water was 78 per cent, a flotation equipment was erected in the expectancy of raising the recovery to over 90 per cent (Fig. 328).

At the Utah-Leasing, Newhouse, Utah, a tailing dump containing about 700,000 tons assaying about 0·7 per cent of copper was successfully treated by flotation, the concentrate assaying about 18 per cent and the final tailing about 0·2 per cent.

At the Calumet and Hecla, Lake Superior, the slime resulting from the stamp-crushing of conglomerate ore containing native copper is treated by a flotation plant having a capacity of about 2000 tons per day. From this material, which contains less than 1 per cent of copper, a recovery 60 per cent is made at a cost of about 10d. per ton. It is proposed to regrind the sand tailing for flotation.

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[1] *Trans. A.I.M.E.* Vol. LV., 1917, p. 620.

At the Britannia, British Columbia, flotation participates with sorting and water-concentration in achieving a recovery of about 94 per cent from a copper ore assaying 2·75 per cent, at a total cost of 2s. 6d. per ton. There are four milling units, each of about 500 tons daily capacity (Fig. 330).

At the Miami, Arizona, 40 per cent of the ore is treated by water-concentration only, 30 per cent by water-concentration and flotation, and the remaining 30 per cent by flotation only.

At the Calaveras-Copper, California, where chalcopyrite is associated with much barren pyrite and the ore assays about 2·15 per cent of copper, flotation recovers 95 per cent in a concentrate assaying 14 per cent; with a capacity of about 200 tons per day, the cost is about 2s. 3d. per ton. Water-concentration had previously only been able to make a 50 per cent recovery in a 6 per cent concentrate.[1]

The capital cost of the National-Copper flotation mill in the Coeur d'Alene district, Idaho, with a capacity of 500 tons per day, was at the rate of £60 per ton of daily capacity. This mill treated ore assaying 0·8 per cent of copper, and produced a tailing assaying 0·02 per cent.

At the Utah-Copper, flotation replaced vanners in the treatment of slime; it is also used to clean a sand concentrate made by vanners.

At the Engels-Copper mine, Plumas County, California, using all-flotation and treating about 800 tons per day, the recovery is about 83 per cent from an ore assaying 2·2 per cent. The concentrate assays about 30 per cent and the tailing 0·45 per cent. The entire cost is about 4s. per ton and the power consumed 22 k.w. hour per ton (Fig. 326).

At the Old-Dominion, Globe, Arizona, treating 800 tons per day, about 47 per cent of the recovery is obtained from jigs, 30 per cent from tables and vanners, and 23 per cent from flotation.

Copper-Gold.—At Mount Morgan, Queensland, where the ore is an auriferous iron and copper pyrite, that portion containing more than about 53 per cent of silica is concentrated by flotation, the remainder being smelted direct. Tables ahead of flotation take out the iron pyrite. For flotation the ore is ground dry in ball-mills to 50 mesh.[2]

At the Falcon mine, Rhodesia, treating an auriferous chalcopyritic ore assaying 2·5 per cent of copper and 5 dwt. of gold, pyrite and pyrrhotite being present, water-concentration recovers about 60 per cent of the gold and 20 per cent of the copper in a concentrate assaying 5 per cent copper and 20 dwt. of gold, while flotation recovers about 20 per cent of the gold and 70 per cent of the copper in a concentrate assaying 8 per cent of copper and 5 dwt. of gold, the total recoveries being 80 per cent of the precious metal and 90 per cent of the copper. The capacity of the equipment is about 500 tons per day; water-concentration is the first treatment and flotation the second; flotation treats about three-quarters of the ore.[1]

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[1] Robbins, *M. & S.P.*, Nov. 25, 1916, p. 772.

[2] Shellshear, *Proc. Aus. I.M.E.*, Nov. 22, 1916.

Gold.—At Cripple Creek, Colorado, where the ore contains auriferous tellurides, some pyrite, and other sulphides, in a gangue consisting of quartz, fluorite, and dolomite, experience has shown that gravity concentration followed by cyanidation makes the best combined treatment; flotation though tried was not favoured, largely because of difficulty in satisfactorily selling the resultant concentrate.

On the Mother Lode, California, where the standard treatment is amalgamation followed by vanners or blankets, the tailing being largely unamenable to cyanidation by reason of cyanicides present, flotation has received some application. At the Belmont-Shawmut, where no amalgamation is practised, roughing tables take out the coarse and flotation the fine pyritic sulphides, these sulphides, largely pyrite, being present to the extent of about 7 per cent (Fig. 336).

The ore at the Belmont-Surf Inlet mine, British Columbia, an auriferous quartz containing unimportant amounts of copper and silver, is crushed to 20 mesh and tabled, the tailing after further comminution being reground and then floated.

At the Oneida-Stagg, Idaho Springs, Colorado, treating a gold ore, cyanidation was displaced by flotation.

Silver.—At Guanajuata, Mexico, experimentally, flotation gives satisfactory recoveries from the fresh siliceous silver ore but not from ‘ fills ’ or decomposed ore; the difficulty is to market the flotation concentrate. Generally speaking, this class of silver ore is better treated by cyanidation, though when the silver occurs as fairly granular argentite flotation may be considered. For such argentite ores, where cyanidation would cost 6s. per ton, flotation would cost about 3s., to which latter figure must, however, be added the extra cost of marketing.

Rose, at the Santa Gertrudis, El Oro, Mexico, found experimentally that flotation in an alkaline circuit made a good recovery from the gold-silver ore, and experienced no difficulty in cyaniding either the flotation concentrate or the flotation tailing. Flotation recovered 65—70 per cent of the precious minerals, chiefly argentite, effecting an enrichment of about 60 : 1, whereas water-concentration would only have made a recovery

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[1] Adam, *Jour. Chem. Min. Met. Soc. of S. Africa,* Nov. 1916.

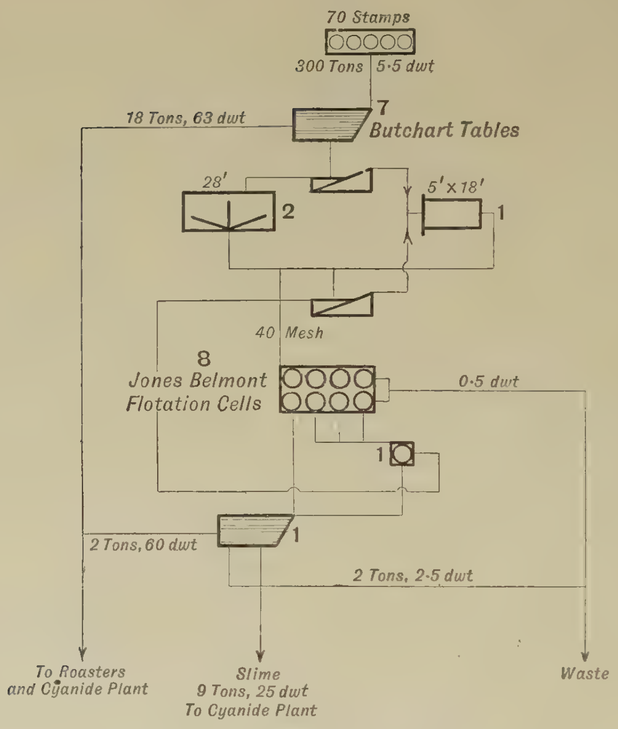

Fig. 336.

Flotation at the Belmont-Shawmut, Tuolumne, California.—Flow-sheet. Auriferous pyrite in quartz and schist; low-grade Mother-Lode ore, with very little free gold. Primary water-concentration, secondary flotation. Recovery about 90 per cent. Flotation concentrate requires to be cleaned by tables because of the talcose material which floats with it. Oil used is a mixture of three parts of wood-creosote and four parts of fuel oil; mixture added in tube-mill; amount added, 1·75 lb. per ton. Sodium sulphide is added to the extent of about 0·2 lb. per ton, partly to the pulp entering the cleaner cell and partly in the tube-mill circuit; this chemical is vital to good work (pp. 435, 483).

The total power-consumption in this concentrator is 19 k.w. hour per ton of ore, the various operations participating in the following proportions:

Crushing and Conveying	6·8	per cent
Stamping	49·8	,,
Tube-milling	17·1	,,
Tabling	2·6	,,
Elevating and Separating	6·0	,,
De-watering Concentrate	1·7	,,
Floating	14·9	,,
Lighting	1·1	,,
	100·0	,,

(Parsons, *M. & S.P.*, November 6, 1920, p. 660.)

of 10 per cent; cyanidation of the raw flotation-concentrate effected from that concentrate a recovery of 90 per cent in the form of bullion.[1]

Elsewhere in the same El Oro district, using sodium sulphide to assist the flotation of the oxidized ore, flotation followed by cyanidation gave a recovery of 86 per cent at a cost of 14s. per ton, whereas straight cyanidation, while it gave the same recovery, cost 16s. per ton.

Tests by Atckison to float oxidized silver ores did not encourage the change from cyanidation; his experience was that the cyanide consumption of the flotation tailing was as high as that of the raw ore, and the solutions fouled quickly.[2]

At Cobalt, Ontario, flotation has largely displaced water-concentration in the treatment of fine sand and slime, though jigs and sand tables continue to produce the large proportion of the concentrate. The flotation concentrate is relatively poor, and for that reason difficult to market. The material submitted to flotation assays about 6 oz. of silver, the concentrate about 300 oz., and the tailing about 1·5 oz.; concentrate obtained by the water assays about 1000 oz. per ton. All the silver minerals in the ore, argentite, ruby silver, and metallic silver, are readily floated. At some mines cyanidation takes the place of flotation. At one mine, flotation following cyanidation was given up, water-concentration taking its place.[3]

When floating a native-metal ore containing only a small amount of sulphide mineral the concentrate unavoidably will contain a large proportion of gangue, a greater enrichment is not possible without making an unmarketable middling or a high-value tailing. Del Mar[4] mentions a silver mill where the feed assayed 11 oz., the tailing 1·5 oz., and the concentrate about 1200 oz., and where native silver and subordinate polybasite were the ore-minerals.

Miscellaneous.—At the Butler tin-wolfram-lead mine, Torrington, New South Wales, galena is separated by flotation from cassiterite and wolframite, these three minerals occurring together in a water-concentrate assaying 53 per cent of lead, 12 per cent of tungstic acid, and 5 per cent of lead; the cassiterite and wolframite are then separated magnetically.

At the Burma Queensland Corporation, a bismuth-wolfram-molybdenum mine in Queensland, the bismuth and wolfram are recovered by water, while the molybdenum is recovered by flotation. By screening the molybdenite froth over an 80-mesh screen an oversize assaying 85—94 per cent of MoS_2 is obtained, and an undersize the best portion of which is kieved to 80—85 per cent MoS_2, while the poorer portion is returned to the flotation system.[5]

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[1] *Trans. A.I.M.E.* Vol. LV. p. 432, 1916.

[2] *M. & S.P.*, April 27, 1918, p. 575.

[3] Callow and Thornhill, *Trans. Can. M.M.I.*, Mar. 1917.

[4] *M. & S.P.*, Oct. 8, 1921, p. 498.

[5] Bowater, *Proc. Aus. I.M.M.*, No. 40, 1920, Abst. *M. Mag.*, Sept. 1921, p. 182.

At the Dominion molybdenite mine, Quyon, Quebec, with molybdenite disseminated in quartz diorite, the mill ore assays 0·5 per cent to 0·75 per cent MoS_2. Treated by flotation the rougher concentrate assays 10 per cent to 15 per cent MoS_2, and contains about an equal amount of pyrite; the cleaner concentrate assays 60 per cent to 70 per cent, this figure being eventually brought to 90 per cent by sizing on an 80-mesh screen.[1]

Cinnabar has been floated in an experimental mill, a concentrate assaying 5 per cent of mercury, and a recovery of 80 per cent being obtained from an ore assaying 0·15 per cent.[2]

To-day flotation is applied on practically every important non-ferrous mining field; on some fields more, on others less. Starting from a successful application to zinc, it became applied successively to the other major base-metals, its most striking capture being the large disseminated-copper deposits. Tin alone has not yet benefited by it. It has largely taken from water-concentration the fine treatment of base-metal ore while it tilts at the ascendency of cyanidation in treating precious-metal ore.

At the expense of water-concentration flotation has progressed not only because of the improved recovery which it brings, but also because of the low cost and the high capacity of its machines, these factors so reflecting upon the working cost that, in spite of the necessity to grind finer and the greater consumption of power, the operating cost of flotation is much the same as that of water-concentration (p. 478). Where flotation is added to treat fine material formerly run to waste, the total cost of treatment will naturally be somewhat increased, but that increased cost is abundantly made good by the improved recovery. This improvement in recovery varies with the extent to which flotation is applied, and with the particular ore; in general terms, where formerly water-concentration yielded a 65—70 per cent recovery, the adoption of flotation might well raise the recovery to 80—85 per cent.

Against cyanidation or chemical treatment generally, flotation has not progressed, except in the direction of removing such base minerals as would interfere with cyanidation, or of recovering such as might otherwise be lost; some precious ores contain sufficient soluble copper to make cyanidation impossible, some contain sufficient galena to be valuable if readily recoverable. Where gold and silver ores are amenable to cyanidation, there is little field for flotation, which, at best, produces a rich concentrate, whereas cyanidation produces bullion (p. 480).

Flotation now treats tens of millions of tons a year, but water-concentration, working at the coarser end of the treatment and treating probably an even greater tonnage, produces more concentrate.

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[1] Oliver, *E. & M.J.*, April 10, 1920, p. 840.

[2] Stowell and Coghill, *M. & S.P.*, Jan. 24, 1920, p. 117.

The Bases of Flotation

Wetting and Non-wetting by Water.

Any explanation of flotation begins with considerations of the phenomenon of wetting. Wetting is the spread of a liquid over a solid to the displacement of air or of another liquid. Ordinarily, wetting is the spread of water over a solid surface to the displacement of air. Some substances, such as clean glass, are readily wetted; upon others, the greasy cabbage leaf, for instance, water does not spread but stands in drops. Similarly, and speaking generally, gangue and oxide minerals are readily wetted, whereas sulphide minerals appear to keep the water from spreading.

Contact Angle.—Upon a readily-wetted surface, water spreads as a thin layer, the surface of which at its confines curves sharply down to make contact with the solid surface at an acute angle (Fig. 337). Upon a greasy surface the water drops raise themselves against gravity till the contact angle may no longer be acute but obtuse. This phenomenon is a display of interfacial tensions or, reciprocally, of interfacial adhesions.

Interfacial tensions result from the unbalanced molecular attractions at surfaces. Surface, being the seat of potential energy, tends to contract and bring thereby this energy to a minimum; hence surface tensions act along the surface. Adhesion, being molecular attraction across the interface, creates a pressure, in response to which a mobile liquid moves along that face; capillarity is an expression of adhesion. When adhesion is high interfacial tension is low; one may be regarded as the reciprocal of the other.

The contact angle marks the equilibrium between the surface tension of the water, σ_1, that of the solid, σ_2, and that of the interface between water and solid, σ_{12} (Fig. 337). Each of these three forces tends to contract the extent of the particular surface from which it derives. The solid being rigid, neither the free solid-surface nor the interface can be diminished by actual contraction of the solid; virtual contraction results, however, when the point of contact is appropriately displaced along the solid surface. That being so, it is not difficult to see that the extent of wetting is primarily determined by the relation between the solid surface-tension, σ_2, and the interfacial tension, σ_{12}. If the solid surface-tension be the greater, the water is pulled over the solid, till such an acute contact angle, θ, is reached that the component of the water tension is a sufficient aid to the interfacial tension to bring-about equilibrium; under these conditions it may be said that the solid displays a preference for water. On the other hand, if the interfacial tension be the greater, the water will be drawn back, the layer will be thickened, and an obtuse angle will obtain; in other words, the solid will display a preference for air.

The equilibrium reached in wetting is expressed by the formula

$$\sigma_{12} + \sigma_1 \cos \theta = \sigma_2 ; \text{ or, } \sigma_1 \cos \theta = \sigma_2 - \sigma_{12}.$$

This formula, while expressing the only available quantitative relation between the different tensions, does not permit the determination of the solid/air tension nor that of the solid/liquid tension, since it contains two unknowns, those two identical tensions; the other factors, σ_1, the liquid tension, and θ, the contact angle, are either known or capable of direct measurement. It gives, however, in the form $\sigma_1 \cos \theta$, the difference between the solid tension in air and that in water.

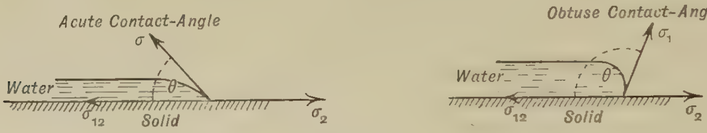

FIG. 337.

The Contact Angle of Wetting.—Diagrams. The contact angle is that between the solid/liquid and the liquid/air surfaces; it is the water angle at the point of contact. An acute angle indicates a tendency to wetting and a preference for water; an obtuse angle a tendency to non-wetting and a preference for air. With an acute angle the solid tension, σ_2, is powerful enough to draw the water over the solid; with an obtuse angle the interfacial tension, σ_{12}, draws the water back from off the solid. The water tension, σ_1, directs itself to bringing-about an equilibrium. θ is the contact angle (p. 487).

Since this difference, $\sigma_1 \cos \theta$, measures the degree of wetting it is interesting to note its variations as the contact angle increases from 0° to 180° (Fig. 338). When the contact angle is 0°, $\cos \theta = 1$, and the degree of wetting is represented by its maximum value, infinity; wetting is complete. When $\theta = 90°$, $\cos \theta = 0$, and there is no tendency, at least none resulting from molecular forces, for the water either to spread or to contract, the spread of water or the degree of wetting may be regarded as unity. Finally, when $\theta = 180°$, $\cos \theta = -1$ and there is a tendency for the water to contract its extent, wetting may then be said to be zero. Accordingly, wetting decreases continuously, and non-wetting increases, from a contact angle of 0° to one of 180°. In other words, and as is exposed by the following statement, wetting is complete when the solid tension is equal to or greater than the sum of the others, while non-wetting is complete when the interfacial tension preponderates similarly.

$$\sigma_1 \cos \theta = \sigma_2 - \sigma_{12},$$

whence $\sigma_2 = \sigma_{12} + \sigma_1 \cos \theta$; and $\sigma_{12} = \sigma_2 - \sigma_1 \cos \theta$

$\theta = 0°$, $\cos \theta = 1$ $\sigma_2 \geqq \sigma_{12} + \sigma_1$ $\sigma_{12} \leqq \sigma_2 - \sigma_1$
(complete wetting).
$\theta = 90°$, $\cos \theta = 0$ $\sigma_2 = \sigma_{12}$ $\sigma_{12} = \sigma_2$
$\theta = 180°$, $\cos \theta = -1$ $\sigma_2 \leqq \sigma_{12} - \sigma_1$ $\sigma_{12} \geqq \sigma_2 + \sigma_1$
(complete non-wetting).

Under ordinary conditions and without designed contamination of the solid, a contact angle as high as 90° between minerals and water does not often obtain, but with contamination 90° may be somewhat exceeded; the contact angle between water and insoluble oil is greater and may

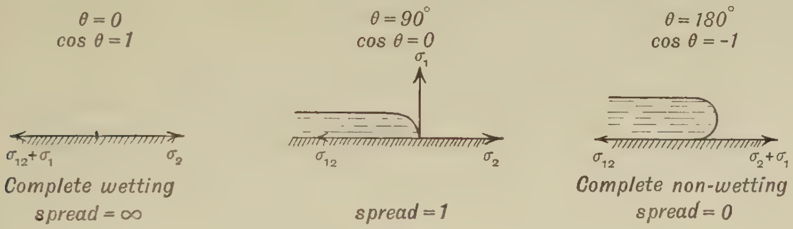

FIG. 338.

Variation of Wetting with Increase of the Contact Angle.—Diagrams. With complete wetting the spread of the water continues till the surface is covered, or till the substance of the water will permit no finer film. Though a limited surface may thus be completely wetted, the imperfections of solid surfaces do not usually permit such wide spread. On the other hand, the spread of some oils upon water is only limited by the impossibility of the film to be less than one molecule in thickness. Unit spread is taken to be that compelled by gravity (p. 488).

be 180°, as will be discussed later. On the other hand, with clean glass and sometimes with quartz, the contact angle with water is appreciably zero, that is to say, the water surface is tangent to the glass surface; this tangency of the two surfaces is, for instance, the assumption in the determination of the surface tension of water by capillary rise.

Hysteresis of the Contact Angle.—The contact angle described so far is that normally assumed at the water edge as spreading water comes to rest upon a plane solid-surface; it is an angle which may be calculated from the depth and area of a water drop upon a solid surface.[1] But before the water comes to rest larger angles may be measured. With gravitational or other outside force causing the water to advance, the point of contact does not shift continuously but in minute jerks, at each of which the attachment of the water surface to the solid surface is broken; just before this break a larger contact angle obtains. Similarly, as water retreats, the water surface is pulled back and a smaller contact angle may be measured before the anchor lifts. In other words, water does not move forward to take up a new position until a certain maximum contact angle, θ_1, has been exceeded; nor will it retreat before a certain minimum angle, θ_0, has been passed. The angle between these two extremes has been described by Sulman[1] as the 'hysteresis' of the contact angle.

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[1] Langmuir, Faraday Society, July 14, 1919. Abst. *M. & S.P.*, Dec. 25, 1920, p. 913.

Hysteresis is well seen when a flat surface having a drop of water upon it, is slightly tilted; then, before the drop runs off, the contact angle on the lower side of the drop will assume its maximum angle, while that on the upper side will assume its minimum (Fig. 339). The fact that both upper and lower contact angles suffer this change indicates that the water surface is anchored against movement in either direction, and suggests that, quite apart from the adhesion between the water and the solid, there is a definite attachment of the water surface to the solid surface. Seeing, however, that still larger contact-angles are formed when the water drop comes to a boundary of the solid surface—a matter discussed later—it may be that hysteresis is due to minute imperfections in the solid surface, such as might throw the solid tension and the interfacial tension out of plane, in which position the water tension, having only to balance the diminished components of those two tensions, might stand more normally to the solid surface. Certainly, though a shake will cause average conditions to obtain and the average contact-angle at once to be taken up, hysteresis of the contact angle appears to be something more than the lag of effect after cause.

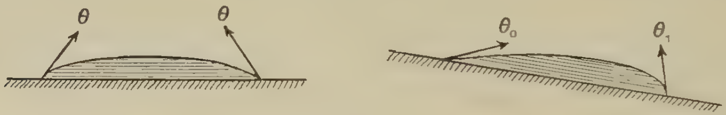

FIG. 339.

Hysteresis of the Contact Angle.—Diagrams. Display of hysteresis upon tilting a plate having a drop of water upon it. θ_0 represents the minimum contact angle, θ, the ordinary, and θ_1 the maximum contact angle. The hysteresis is accordingly represented by $\theta_1 - \theta_0$ (p. 489).

The contact angle is also observable when a horizontal water-surface meets a vertical solid-surface (Fig. 340); water in a glass vessel climbs the sides to form an acute contact-angle, or one which, when the glass is clean, may be zero; mercury, on the other hand, as in the ordinary vertical barometric tube, gives an obtuse angle. Under such circumstance, though the general character of the angle may be appreciable by the eye, the angle itself is difficult of precise measurement. This difficulty is contoured by dipping the mineral specimen, on which a plane surface has been prepared, vertically into the water, and then laying it over till the meniscus disappears and the horizontal water-surface continues right to the mineral surface. The air angle, $180° - \theta$, is then readily measured, whence θ, the contact or water angle, is obtained by subtraction from 180°. If, now, the mineral surface be moved back toward the vertical position, the meniscus does not begin to re-form until a certain angle has been moved through, hysteresis again displaying itself. With representative sulphides the hysteresis angle thus disclosed shows itself generally to be a little more than equal to the minimum contact angle; that is to say, the maximum contact angle is a little greater than twice the minimum. When, however, the sulphide surface has been oiled the maximum is greater, and generally about three times the minimum.[1] This last fact is interesting in that it is explicable rather upon the hypothesis that hysteresis is the expression of attachment or 'drag,' rather than of surface irregularities such as the oil film might be expected to make smooth.

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[1] *Trans. I.M.M.* Vol. XXIX., 1920, p. 88.

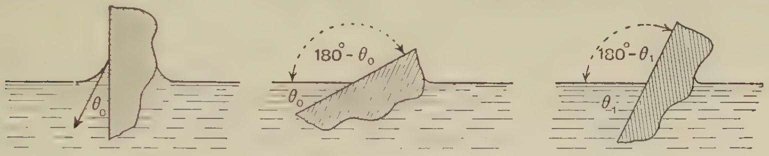

Fig. 340.

Measurement of the Contact Angle and the Hysteresis Angle.—Diagram. It is difficult to measure the contact angle when the water surface merges into a curved meniscus, as against vertical and horizontal surfaces. However, by laying the surface of the mineral over, till the meniscus disappears, the contact angle becomes that between the horizontal water surface and the inclined mineral surface. In that position the air-angle, $180° - \theta_0$, is measured and the water angle obtained by subtraction; this angle would be the minimum contact angle. If, now, the mineral surface be turned back again towards its original position, the meniscus does not begin to re-form until a certain angle, that known as the hysteresis angle, has been traversed; the contact angle measured in this position would be the maximum, θ_1 (p. 490).

Contact angles and hysteresis are, accordingly, the factors in wetting; the former is determined by the relation between the interfacial tension and the solid tension; the latter is a measure of the anchoring of the water surface to the solid surface. It goes without saying, that these factors obtain also with small solid particles on water, though it is probable that they are affected by the relatively-greater surface energy of such particles.

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[1] Sulman, *Trans. I.M.M.* Vol. XXIX., 1920, p. 114.

Suspensory Conditions and Floatability

Though a contact angle above 90° expresses a degree of non-wetting and a preference for air, it does not express repulsion for water; at all possible contact angles, however great, there is still adhesion between the mineral and water, that is to say, still a force which, apart from gravity, would hold a particle down to a water surface; such adhesion is manifested, for example, by the power of even a dry piece of galena upon being withdrawn from depression into water, to lift an amount of attached water appreciably above the water-level (Fig. 341). Reciprocally, though a contact angle less than 90° expresses a degree of wetting it does not express repulsion for air; at all possible contact angles, however small, there is still interfacial tension, that is to say, still a force tending to hold a particle to a water surface; witness the fact, referred to again later, that relatively large pieces of glass having a zero contact-angle, are readily floated.

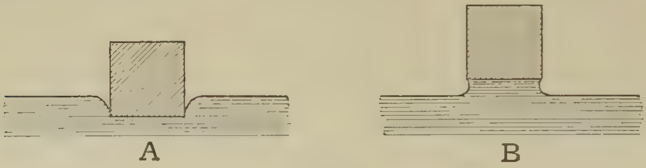


Fig. 341.

Adhesion between Water and Mineral.—Diagram. A shows a contaminated cube of galena depressed into water and forming a contact angle greater than 90°; B shows the same cube lifted out of the water again, having brought with it an amount of attached water. This lifting of the water indicates adhesion across the liquid/solid interface, though the attachment of which hysteresis is evidence is partly responsible (p. 492).

It is therefore evident that contact angles and hysteresis, though indicative of the ease or difficulty of wetting and, as will be described later, some guide to floatability, do not directly disclose the essential relations in flotation; the same may be said of wetting itself. There is a further position to be considered.

Edge Angle.—Returning for a moment to discuss water moving over a solid, the final position taken up by the water surface is at the edge of the solid. At such an edge the behaviour of the water suggests that the solid tension, now acting around the corner, is out of action upon the point of contact; it is as though the solid tension were absent. Under those conditions two tensions only remain to determine the contact angle, the interfacial tension, σ_{12}, and the water tension, σ_1. Where the former of these two was the greater, as it usually is, and where gravity did not disturb the position, the water surface would move right over to oppose its full tension to the interfacial tension, at an 'edge' angle, θ_2, of 180° (Fig. 342). With an interfacial tension less than the water tension, the water surface would only move over till the edge angle were such that the water component along the interfacial plane balanced the interfacial tension.

The edge angle is well seen when a glass is filled above the brim, as it may be; then, even though the contact angle were acute all the way up the side, arrived at the top an edge angle approaching 180° is formed (Fig. 342). It is also indicated by the manner in which the contact angle builds up along the edge of a plate over which water is flowing, or on to which a water drop is hanging (Fig. 342); in this latter situation the attachment is ruptured earlier than 180°, and at an angle nearer 90°, owing to pressure due to gravity.

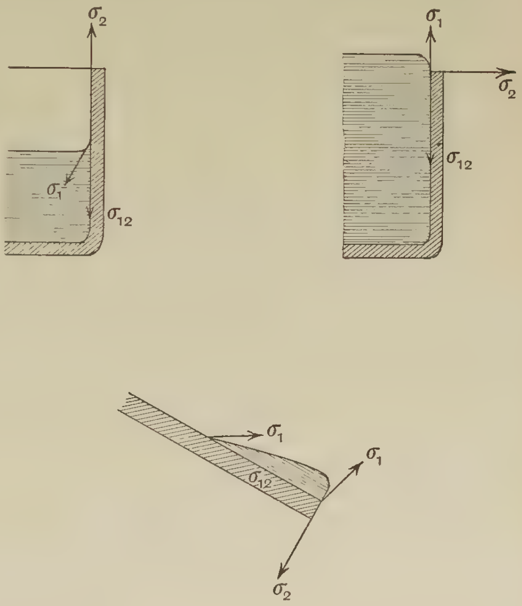


FIG. 342.

Edge Angle.—The edge angle is that contact angle which builds up when water reaches a solid edge. In that situation the solid tension no longer acts in the same plane as the interfacial tension, and no longer enters the equilibrium which ordinarily determines the contact angle. The edge angle is seen when a glass is filled above the brim; it is also suggested by the way water hangs to the edge of a glass plate (p. 493).

Suspensory Angle and Suspension. — When spreading water comes to rest upon a horizontal solid surface and a contact angle, θ, obtains, the curved meniscus which holds the water in check tends to exercise a lifting effect upon the solid at the point of contact; the measure of this lifting effect would be $\sigma_1 \sin \theta$ per unit length of contact, and $l\sigma_1 \sin \theta$ for any length, l, of contact. The circumstance of a particle upon a water surface is a little different; contact with the water is first established at the side and not upon the top, the contact angle will accordingly obtain at the side (Fig. 343). If the normal contact angle be less than 90° the vertical component of the water tension, $\sigma_1 \cos \theta$, will act with gravity to draw the particle down, being under this circumstance a sinking component. Hysteresis, on the other hand, will tend to support the particle; adhesion and drag accordingly here tend to act oppositely, and only if hysteresis be the greater will there be any force to set against gravity. When the particle, which for convenience may be taken to be a cube, has sunk so far that its upper surface is at water level, the molecular sinking effect is extinguished and a supporting effect begins, adhesion and drag afterwards acting together. Sinking deeper, an edge angle forms. If, now, the interfacial tension be greater than the water tension, and the particle be heavy enough, the water surface is pulled down till an edge angle, θ_2, of 180° obtains, this giving a 'suspensory angle,' $\theta_2 - 90°$, of 90°—the suspensory angle is the angle of the depressed water surface to the horizontal; in this position the two tensions, acting oppositely, are in the same plane, and the water is making its maximum suspensory effort, namely, that represented by its full surface tension. So long as the particle is just held by this maximum effort, the point of contact does not turn the corner; on the other hand, any tendency for this point to be moved downward by the superior force of the interfacial tension is at once corrected by the solid tension resuming action. When the particle is too heavy to be so held, the water turns the corner, and the particle is overwhelmed and sinks. In thus being overwhelmed, if the contact angle and the hysteresis of the encroaching water together be substantially more than 90°, it is likely that the particle will sink with an air bubble attached.

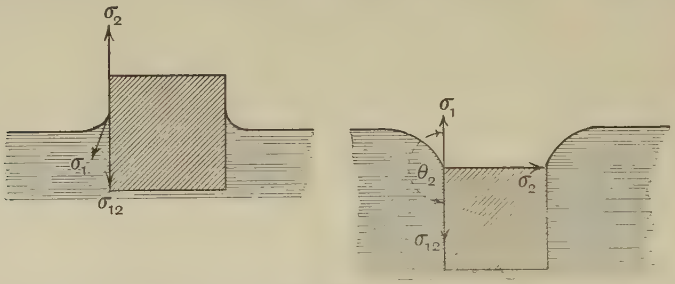


FIG. 343.

Suspension of a Large Particle.—Diagram. The left-hand drawing indicates the position as the particle sinks into the water and momentarily displays its minimum contact angle, in this case an acute angle. In the right-hand drawing the floating position of the same particle is shown, the case taken being one where the interfacial tension σ_{12} is greater than the water tension σ_1; and consequently where the edge angle θ_2 is 180°, and the suspensory angle, $\theta_2 - 90$, is 90° (p. 493).

If the interfacial tension be less than the water tension and the particle be appropriately heavy, the water surface is pulled down till the suspensory angle is such that equilibrium obtains between the vertical component of the water tension and the interfacial tension. The suspensory effort then put forward by the water is $\sigma_1 \sin a$, a being the suspensory angle. Where the particle is too heavy to be held by this effort the water encroaches and, as before, the particle sinks.

Applying these reasonings to ordinary crushed particles which, though not put forward as cubes, provide at their corners and in the imperfections of their surfaces the necessary edges for attachment, it will be seen that when the interfacial tension of a particular particle is greater than the water tension the suspensory effort is at a maximum, and that this maximum is represented by σ_1, the water tension. But that when the interfacial tension is the lower of the two tensions this effort is represented by σ_{12}, the interfacial tension itself. So long, therefore, as interfacial tension exists, there is some tendency to float. All tendency to float is only destroyed by reducing the interfacial tension to zero. Zero contact-angle has not the same significance; it is easy to float relatively large particles of clean glass upon acidified water, though the contact angle then is zero. Zero contact-angle only signifies that under the conditions obtaining the interfacial tension is at a minimum; on the other hand, when the contact angle is greater than zero, in that fact alone there is evidence that interfacial tension exists.

On the foregoing assumptions the dimension D of the maximum cube of density δ which would float would be found by equating the suspensory effort to the weight which has to be supported; thus, ω being the weight of unit volume of water:

$$D^3(\delta - 1)\omega = 4D\sigma_1; \text{ or, } D = \sqrt{\frac{4\sigma_1}{(\delta - 1)\omega}}.$$

The same maximum size of floatable cube may be derived from other considerations. It has already been stated that the molecular sinking effect of the contact angle is $\sigma_1 \cos \theta$, the floating effect will consequently be $\sigma_1 \cos (180° - \theta)$. If it were possible for θ to be 180°, the floating effect would be σ_1 and a maximum. But at such a contact angle the water surface would remain free all round and under the cube. The dimension of the maximum floatable cube would then be obtained by equating the work done upon the water in sinking, with the energy in the increased water surface. The total work done in sinking till the upper surface of the cube was level with the water surface would be $D^3\delta\omega D$, of which amount $D^3\omega D$ would represent the gravitational work done in displacing the water, leaving $D^3(\delta - 1)\omega D$ to be equated to the increased surface energy. This increased energy would be that on the total surface fronting four faces of the cube, or $4D^2\sigma_1$, and the formula becomes:

$$D^3(\delta - 1)\omega D = 4D^2\sigma_1; \text{ or, } D = \sqrt{\frac{4\sigma_1}{(\delta - 1)\omega}}.$$

In both of these reasonings the relatively small amount by which, owing to the depression in which the cube floats, the water displaced is greater in volume than the particular cube, has been neglected; if included, the size of the maximum floatable cube would, conceivably, be somewhat increased, though the extra work done in sinking deeper might be considered as absorbed by the greater surface associated with the meniscus. Applying the above value for D, the size of the maximum floatable cube of galena would be about 2 mm., and weigh about 60 milligrams.

With the sphere it is a little different. A perfect sphere would have no edge angle, and theoretically would only float if the contact angle were 180°. Practically, however, the imperfections of the surface provide the drag which, converting a low contact angle to a high one, prevents the sphere from being overwhelmed. Assuming the effective contact angle to be 180°, and the contact to extend around the greatest diameter, the formula for the diameter D of the maximum floatable sphere would be:

$$\frac{\pi}{6}D^3(\delta - 1)\omega = \pi D\sigma_1; \text{ or, } D = \sqrt{\frac{6\sigma}{(\delta - 1)\omega.}}$$

The same value for the diameter would be reached by considerations of the work done and energy transplanted. This value for the diameter would be $\sqrt{1{\cdot}5}$ times greater than the side of the maximum floatable cube; the respective masses of sphere and cube would accordingly be the same. A needle, which may be taken to be a long cylinder, floats perhaps a little above the maximum diameter.

In all the uncertainty it is nevertheless clear that the water surface-tension is a primary factor in flotation; when a solid piece approaching the limit of floatable size has been carefully floated, a slight reduction of the surface tension by the introduction of a contaminant, will cause the piece immediately to sink. Another primary factor is the total length of the contact, the length of the contact-outline; a needle gives a much longer contact-outline than a sphere or cube of the same mass, and for this reason is more easily floated; similarly, flaky mineral floats more readily and persistently than that which is relatively equidimensional. The part played by the length of contact-outline is also well disclosed by the ready flotation of most small particles. While mass diminishes with the cube of the dimension the contact-outline diminishes only in direct proportion to the dimension, the consequence being that the small particle has a relatively large contact-outline. Accordingly, since the total suspensory effort is the product of contact-outline and the vertical component of the water tension, a low suspensory angle may suffice for the flotation of a small particle; indeed with small particles the point of contact may not reach the top of the particle, the particle may float with its top above water (Fig. 344). Though, even then, it is possible that edge angles obtain upon irregularities of the surface, it is probable that the suspensory angle is determined in greater part by the hysteresis, that is to say, its value will be $\theta_1 - 90°$ rather than $\theta_2 - 90°$.

The total suspensory effect upon a number of particles being proportional to the sum of the lengths of the separate contacts, fine material dusted upon a water surface is capable of supporting a relatively great weight. Galena particles smaller than 200 mesh can, for instance, be sprinkled upon a water surface till the weight supported per square centimetre approaches 3000 milligrams, none falling through. All that great weight of galena is borne upon that film of particles which is in contact with the water. Sprinkling but a thin layer of galena over the surface some properties of this mineral film may be noticed: the particles out of contact with the water are carried in a cradle-like depression. A knife-edge thrust into the layer will carry the film downward to form a sack, which, unless the thrust has been too deep, persists when the knife is withdrawn; if thrust too deep, the sack, being too heavy, falls as a plastic coherent mass of irregular shape; in either event the fact is demonstrated, that when once a new surface becomes occupied by floatable particles it does not pass away when the causative force has been removed, but being 'armoured' it continues. Of similar import, if the knife be moved horizontally over the surface to sweep a space clean of galena particles, the particles banked behind the knife will flash back to reoccupy the vacant surface as soon as the knife is removed. Another interesting point is the attraction for one another of particles placed to float freely on the uncontaminated water surface, such particles all running together to form a strong film. On a surface contaminated with a minimum thickness of oil such freedom to move no longer obtains; with a greater thickness of oil, say sufficient to form an interface with the water, it is resumed.

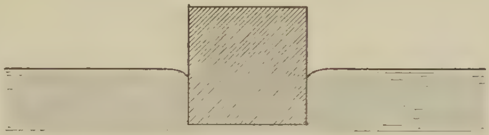


FIG. 344.

Suspension of a Small Particle.—Diagram. In consequence of its relatively great contact-outline, a small particle may float partly above water, the suspensory angle available at the side then sufficing. The suspensory angle there available is $\theta_1 - 90°$, θ_1 being the maximum contact angle (normal contact angle plus hysteresis) (p. 496).

Floatability.—In résumé of the foregoing considerations it may be said that the minerals which the more readily float are non-wettable minerals; non-wetting indicates a relatively high interfacial tension; the greater the interfacial tension in relation to the water tension the more stable the attachment of the particle to the air/water surface and the greater the floatability. To this extent non-wetting is an index to floatability. More precisely, however, non-wetting depends upon the relation between the solid tension and the interfacial tension; floatability, on the other hand, depends upon the relation between the interfacial tension and the water tension. Wetting is the power of water to spread over mineral, displacing air; floatability is the power of mineral particles to attach themselves, under practical conditions, to an air/water surface, and particularly to air bubbles.

The actual value of the interfacial tension, σ_{12}, depends primarily upon the molecular attraction across the interface, in the sense that when that attraction is great the interfacial tension is low, and *vice versa.* Though connected with the solid and the water tensions, interfacial tension is not solely dependent upon those two tensions, since independently of them it may be varied by chemical attack or by changes in electrostatic charge. By raising its value floatability is increased, and *vice versa*; in other words, the floatability of a mineral may be said to depend upon the Surface Energy of the particle in water; it is noted below that, besides mineral character, particle-size is a factor in surface energy.

The difference in behaviour between floatable and unfloatable mineral is well exhibited by comparing galena and quartz. Finely-powdered galena sprinkled upon a water surface can readily be heaped to a depth of 5 mm. before the water surface is ruptured and the galena sinks; powdered quartz of the same fine size, on the other hand, hardly leaves a trace upon the surface but falls immediately through. Again, coarser particles of galena, if dropped on to a water surface so that they break through, take with them each an air bubble, making themselves appear like globules of mercury upon the bottom; quartz particles, on the other hand, sink unaccompanied by air, to lie dull upon the bottom.

It is, however, quite easy to float larger pieces of quartz on a quiet water surface; moreover, it will be found that the quartz pieces possible of thus being floated are substantially larger than the pieces of galena. This difference in behaviour between coarse and very fine particles of quartz is explicable on the assumption that with the latter there is incipient solution, and in consequence such a reduction of the interfacial energy that floatability is destroyed. In this respect minerals are not all alike; very fine cassiterite, for instance, tends to float.

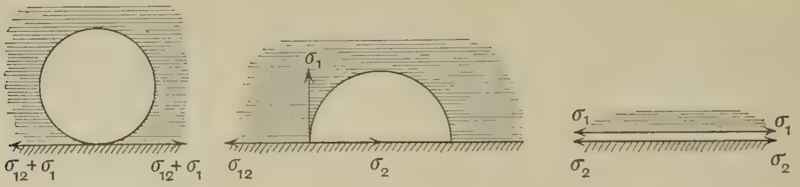
Flotation practice has shown that some minerals are more easily recovered by flotation than others, the following minerals being placed in a rough order of floatability: molybdenite, stibnite, galena, chalcocite, chalcopyrite, sphalerite, pyrite, bornite, etc.; among gangue-minerals there is a tendency for the more basic, such as magnetite, siderite, barite, fluorite, and for those with pronounced cleavage, such as calcite, to float. Much, however, depends upon the condition of the mineral, whether it be fresh or has been long exposed; chalcocite, for instance, is not amenable to flotation when there is long delay between crushing and flotation. On the other hand, crushed ore which has been long exposed, or ore which has been crushed dry, offers difficulties in flotation by reason of the unselective character of the float; with such ore, acid, which possibly cleans the surfaces, is found of particular benefit.

Perfectly fresh minerals are readily wetted; water almost flashes across fresh exposures. But that does not necessarily indicate low floatability; as already stated, the two properties of wetting and floatability are not reciprocals, the one of the other; they depend upon different relations. Experience, however, has shown that wet crushing so completely wets the gangue that any tendency for it to float is diminished if not extinguished.

Previous wetting undoubtedly has a great effect on the contact angle, largely extinguishing it. The contact angle of a floatable mineral immediately after immersion is lower than before, and may be zero. For this reason and because it never obtains under suspensory conditions, the normal contact angle, from the practical point of view, is of little use in flotation. In conjunction with hysteresis, however, it may, as already stated, accomplish the flotation of fine particles (p. 497).

Solid tension is the important factor in wetting but not in flotation. That this tension often appears to be of a lower order than might be imagined from other considerations, is likely due to contamination with air or with gas taken from the air, carbonic acid, for instance. The consensus of opinion is that solids adsorb gases readily; such an adsorption would indicate great surface energy. Contaminated with this adsorbed film they no longer possess their original energy. Freshly-broken surfaces certainly display greater surface tension; they draw the water over themselves, becoming completely wetted. The surface so wetted is one of considerably lower tension; if in water that tension were reduced to zero all chance of flotation would be extinguished, and solution, or, with suspensoid particles, complete deflocculation, would obtain.

Bubble-suspension.—The conditions in respect to wetting and floating which have hitherto been sketched as they obtain on the free surface of standing water, are not radically altered when mineral particle meets air bubble beneath the surface. It is true the mineral then starts wet, making attachment to air a positive operation; but, where the contact angle is greater than zero, quiet contact between mineral and bubble will secure that attachment; where it is zero, impact will empower the particle to pierce the bubble, after which, granting that interfacial tension exists, attachment will follow; that there may be attachment even when the contact angle is zero is manifested by the way air bubbles stick to the sides and bottom of glass vessels containing water. Otherwise, the same molecular forces of the same magnitude come into play at contact; contact angles, hysteresis angles, and edge angles are formed as before (Fig. 345). The same maximum size of particle can be held against gravity to a bubble surface as to the free surface; eventual flotation only requires that the volume of the bubble be such that together with the mineral particle it forms a system lighter than water; rising to the surface the attachment of mineral to bubble has to bear the weight of the particle in water, that is, to withstand the tendency of the particle to sink (Fig. 346). With a smaller bubble the system sinks; the attachment of the bubble to the particle has then only to bear the weight of water displaced by the air, or, in other words, the tendency of the bubble to rise; should, in the end, a sufficiency of other bubbles merge themselves into the attached bubble, the system would reverse its tendency and rise.



$\theta = 0$ $\theta = 90°$ $\theta = 180°$

FIG. 345.

Contact Angles with Air Bubbles in Water.—Diagram. A contact angle of 0° connotes a minimum attachment of an air bubble to a mineral surface (p. 500). At 90° the attachment would be unity because sin θ would be unity and the length of contact unity also. At 180° the attachment would be a maximum, because though sin θ had gradually decreased to zero, the length of the contact-outline had increased at a more rapid rate; the one had become infinitely small, but only after the other had become infinitely large.

In practice, the assembly of mineral and air is generally such that mineral particles form a more or less complete lining to a bubble large enough to float them all; in that way the particles on the side and top of the bubble, needing support in addition to their own adhesion, receive that support from their neighbours; the bubble is said to be ‘armoured.’ Moreover, the particles do not approach the maximum floatable size, since even were it desirable to attempt to float such large particles, the agitation associated with the operation would be incompatible with the quietness necessary for their recovery; the province of flotation is the recovery of fine mineral-particles, and, reciprocally, fine particles bring stability of air bubbles and of froth. Finally, in flotation, some of the mineral particles in the course of agitation are projected to the surface, and have not to wait for their complement of air; once at the surface they find bubbles large enough to support them.

It is possible that when oil is used and the air bubble becomes oiled, the film of oil is continuous over the attached particle, and that, consequently, the particle is floating at an oil-water interface, out of direct contact with the air. It is also possible that air bubble and mineral particle, particularly those of the smallest size, are held together by the attraction of opposite electrostatic charges, in which case, also, the air bubble would remain unpierced (p. 513). Certainly, the general appearance is that the mineral particles are in the water rather than in the air.

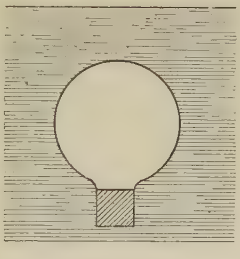


Fig. 346.

Bubble-suspension of a Large Particle.—Diagram. This suspension is the same as at a free water surface; a free surface may, in fact, be considered as the interior surface of an infinitely large bubble. A bubble loaded at the bottom in this way is often described as an 'air-bell.' In present-day flotation complete armouring of the bubble with fine particles is characteristic (p. 500).

Wetting and Contamination by Oil.

Wetting by oil is here discussed not because, ordinarily speaking, it takes place in flotation, but to explain the filming of mineral particles and of air bubbles with oil.

At the contact of oil with a solid in air, equilibrium of the three tensions, σ_0, that of the oil, σ_2, that of the solid, and σ_{02}, that of the interface, obtains at an appropriate contact angle. Oil, however, unlike water, spreads over a sulphide surface, this spreading indicating considerable adhesion—and consequently a low interfacial tension—between sulphide and oil; contrariwise, on a gangue-surface oil, relatively speaking, is restrained from spreading, building itself up to form a greater contact angle (Fig. 347). Similarly, though sulphides characteristically float on water they equally characteristically sink in oil; gangue-minerals, on the other hand, sink somewhat less readily in oil than in water. A similar preference of sulphide for oil is noted at contact under water (Fig. 348).

In flotation a more or less insoluble oil, in small amount yet forming the major part of the oil mixture, is commonly used. This oil being added to the water, the spread of oil on water must be considered. Speaking generally, the oils used spread on water to an extent unaccountable by gravity, often till an iridescent play of colour indicates the extreme tenuity —less than $400\mu\mu$—of the covering film, sometimes even till all visible trace of film has disappeared,—say $15\mu\mu$. Such a spread of oil indicates that the water tension, σ_1, is greater than the oil tension, σ_0, greater also than the interfacial tension, σ_{01}, and often sensibly equal to these two tensions combined (Fig. 349); the water tension is known to be of the order of 75 dynes per centimetre, the oil tension is generally about but less than 50 dynes, leaving the interfacial tension lower than either. The contact oil-angle of these oils upon water accordingly often approaches zero, the contact water-angle approaching 180°. With insoluble oils or oil fractions which do not spread, the oil tension and interfacial tension, while remaining of the same order as before, would be generally somewhat higher, equilibrium then obtaining at a higher oil angle, or inversely a lower water

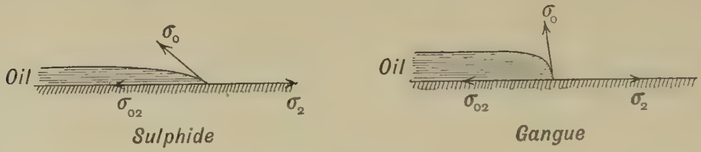


FIG. 347.

Spread of Oil on Sulphide and Gangue.—Diagram. Oil spreads to a greater extent on sulphide mineral than on gangue; in other words, the oil contact-angle on sulphide is smaller than it is on gangue (p. 501).

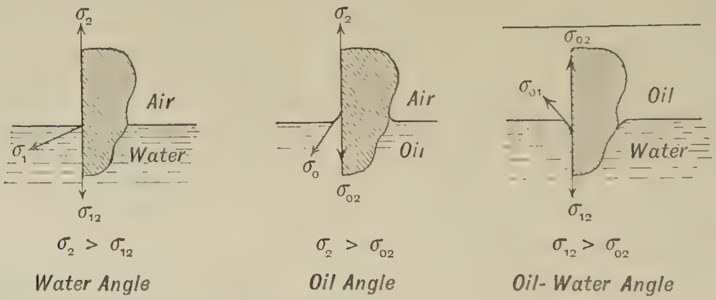


FIG. 348.

Contact Angles of Sulphide with Water and Oil.—Diagram. With water the contact angle between σ_1 and σ_{12} is substantial, but generally less than a right angle; with oil, the oil contact-angle, namely, that between σ_0 and σ_{02}, is less than with water; with oil and water, the water contact-angle, namely, that between σ_{01} and σ_{02}, is greater again, and generally substantially greater than 90° (p. 501).

angle, one substantially less than 180° (Fig. 349). It will be realized that, unlike the position when a solid is being wetted by oil, it is the interfacial tension, and not the tension of the spreading liquid, which by changing the angle at which it acts redresses the balance between the other two tensions.

Extreme spread of oil upon water, though it may be partly due to molecular attraction amounting to fractional solution across the oil-water interface, that is to say, to a low interfacial tension, is chiefly due to the relatively low tension of the oil itself. Similarly, restraint of oil upon water is largely due to higher oil tension, the interfacial tension, while still remaining of a low order, being then more than able to redress the balance.

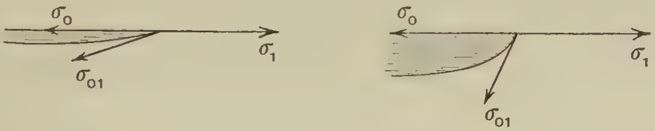


FIG. 349.

Spread of Oil on Water.—Diagram. Contrary to what happens when oil spreads over a solid, when oil spreads over water the interfacial tension σ_{01} does not lie in the plane along which spread takes place; in consequence, the water tension is chiefly opposed by the oil tension, the interfacial tension altering its plane of action to secure equilibrium between the others. When the water tension, σ_1, is practically equal to the other two tensions combined, the water contact-angle between σ_1 and σ_{01} approaches 180°, while the oil contact-angle between σ_{01} and σ_0 is sensibly zero, and the oil spreads as a fine film. On the other hand, when the water tension finds the oil tension a more equal opponent, the interfacial tension aligns itself at a substantial angle to the oil tension, and instead of a film a disc of oil is formed (p. 502).

Mineral Contamination with Oil.—When the oil contains even but an insignificant soluble fraction, then, not long after spreading, the bright film becomes perforated with dull patches which expand till the whole film is involved, when, from out the dullness, bright beads of free oil assume shape by re-assembly at centres, leaving apparently clear spaces between. That these clear spaces are nevertheless filmed with oil is indicated by the inability of further oil to spread over them, and the inability of floating particles to run together. The first of these two phenomena might be taken to indicate either that the surface tension of the water had been lowered and was insufficient to extend the oil film, or that the interfacial tension had increased. The fact, however, that particles are as readily floated upon that surface as upon clean water, leaves little doubt that the surface tension of the water was unaffected, in which case, according to Rayleigh, stiff as it shows itself to be, the film can only be one molecule thick. That the insoluble fraction gathers itself into beads, may be taken to indicate that upon withdrawal of the soluble fraction the oil tension increased and the interfacial tension likewise, both, however, remaining of the same order of magnitude as before.

The selective or collecting oils—the tars, fuel-oils, etc.—characteristically give this stiff, strong, yet scarcely perceptible film on water; those of them which do not spread, the tars, for instance, flash out such a film immediately on contact with water; others, those containing a larger soluble fraction, develop this film after spreading, in the manner described above. Other oils, the insoluble paraffin, for instance, spread in an oily film which does not degradate, and upon which floating particles still retain their freedom to run together.

With increase in the thickness of the film of oil upon a water surface, the water tension gradually becomes less, and at length a normal interface between oil and water becomes established, which, having a tension lower than that of water, cannot support the same-sized particle; some particles which could readily be floated on water will sink in the presence of an oil film thick enough to establish an interface. On the other hand, some particles which would never float on the oil are held at the oil-water interface; the power to hold them there apparently increases as the thickness of the film decreases; at any rate, some particles which quickly fall from a pronounced pool of oil, are held by a tenuous and continuous film (Fig. 350).

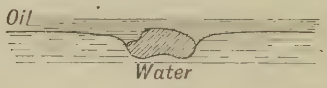


FIG. 350.

Flotation of a Sulphide Particle at an Oil/Water Interface.—Diagram. Though a sulphide particle more readily floats on a water surface than on an oil/water interface, if small enough it may still float at that interface, even after the free oil-surface has been unable to hold it. The inference is that though the oil/water interfacial tension σ_{01} is less than the oil tension σ_0, the interfacial tension of the particle in water is greater than it is in oil (p. 504).

Equally, sulphide particles which drip with oil can hardly be made to float on water. It apparently requires something conceivably approaching a mono-molecular film to increase the floatability of a particle; witness the ease with which a needle may be floated after it has been rubbed between the fingers or passed through the hair.

The greater floatability of a particle properly filmed with oil is due to increase in the interfacial tension between the particle and the water; the tenuous oil-film forms part with the solid, to the extent that while preventing water from coming into contact with the solid to reduce its tension, it does not itself cause a reduction of that tension; nor is it present as free oil. In practice, a sulphide particle once filmed maintains its contamination at least for a second treatment.

Conditions below the surface and in the presence of air bubbles are not radically different from those sketched for the free surface. By agitation, the insoluble oil is broken into a diffusion of droplets, this diffusion resembling but not having the stability of an emulsion. Such droplets, impinging one with an air bubble and another with a mineral particle, spread themselves over the respective surfaces; in turn, the oiled particle and the oiled bubble impinge, to become attached; with sufficient air attached the particle is floated to the surface. The filming of the particle, which is the essential factor, may take place before the introduction of those air bubbles which eventually cause this flotation; seeing, however, that even then the necessary agitation takes place in the presence of air, there is little doubt that some air bubbles are filmed concomitantly with the particles; it is even likely that the filming of these bubbles assists in the filming of the particles, since an unfilmed particle, projected through a filmed air-water surface, would take the film with it.

It may here be remarked that though adsorption would tend to move the oil droplet to the bubble and mineral surfaces, and the mineral particle to the bubble surface, such movements would take time even were the droplets and the particles all of colloidal size. Seeing, however, that in greatest part they are of a larger order of size, that the opportunities for impingement independently of adsorption are numberless, and that upon impingement the respective tensions act quickly, adsorption can hardly play any primary part either in filming or in air attachment.

Amount of Mineral Contaminant.—The amount of oil present should at a minimum be sufficient to film all the mineral particles, due allowance being made that much will go to line numberless air-bubbles. To ensure this filming in reasonable time, the agitation must be relatively violent and the amount of oil greater than the theoretical requirement. Practice sanctions such an amount as calculated over the mineral particles alone would give a film having a thickness of 150–250$\mu\mu$, this amount being generally a small fraction of 1 per cent of the weight of the ore. Below a maximum of about 1 per cent and presuming an adequate emulsion, the greater the amount of oil the greater the recovery possible, since the filming would be more complete; the cost of the extra oil and of the greater agitation necessary to keep it divided would, however, generally more than offset any small extra recovery.

The limiting amount of oil is that which, under the conditions of agitation obtaining, would render possible the formation of an oil film of such thickness that a definite interface between oil and water would be set up and normal air-attachment be prevented. Such oil-enveloped mineral particles, once they had overcome the buoyancy of the oil, would sink, either singly or in the floccules which their common oily surface permitted to assemble, taking the oil with them. In the Cattermole process of concentration by oil, this selective flocculation and sinking was obtained by employing, under gentle agitation, an amount of oil equal to about 2 per cent of the weight of the ore.

Finally, oil might be in such large amount and the agitation so gentle that no separate droplets would be formed, but the oil would remain as a magma, which, working through the pulp, would catch the mineral particles within its mass, and, with the assistance of such air bubbles as were attached to the mineral, carry them to the surface. For this to happen the amount of oil would have to be more than the weight of the ore. This was the practice in the original Oil Flotation process of Elmore.

Obviously, in present-day flotation and using the small amount of oil stated, the degree of oil division or emulsification is an important factor. With complete emulsification no droplet or bead could form at all, nor could the necessary filming of the mineral be achieved; such emulsification must be avoided, and any tendency towards it must be corrected. Again, flocculation of the pulp would tend to carry oil droplets down with the settling slime, and to that extent remove the oil; the pulp should, accordingly, be in a deflocculated condition. Oil, completely emulsified, tends to flocculate slime and to give a poor voluminous float; properly-divided free oil, on the other hand, gives a rich selective float. With pneumatic machines the previous mixing must be so complete as to last through the subsequent float. With mechanical machines, on the other hand, agitation and oil-division are continually renewed, almost to the extent that excessive division may supervene.

Chemical and Electrolytic Action.

To increase Interfacial Tension.—The filming of a particle with oil conduces to non-wetting and to flotation, the interfacial tension in water being thereby maintained higher than were the particle naked. Generally speaking, the oxidized and the oxide minerals do not take this film, their interfacial tension remains low and their adhesion high. To attempt the recovery of these minerals by flotation the mineral surface must be prepared otherwise; something other than oil must be interposed to weaken their adhesion to water, or reciprocally, to increase their interfacial tension in water.

Thus, the oxydized ores of copper, for instance, may be rendered floatable by the addition of an alkaline sulphide, when, either chemically or electrolytically, the metalliferous particles are filmed with an imperceptible film of copper sulphide. Similarly, the oxide of tin, for instance, may be floated by the addition of copper sulphate in the presence of metallic zinc or iron, when the cassiterite becomes filmed with metallic copper (p. 453). To cover the particles in this way with a metallic film has been described as 'metallizing,' a parallel term to 'sulphidizing.' Though such a method is not practised for the recovery of cassiterite, copper sulphate is often used in the flotation of zinc blende, the presence of this electrolyte conceivably resulting in the incipient filming of the blende with copper sulphide.

Acting as an electrolyte, and conceivably resulting in the adsorption of positive ions upon the mineral surface, sulphuric acid has sometimes, as at Broken Hill and elsewhere, been essential to flotation, the sulphides not floating without it.[1]

These particular means to increase or maintain the interfacial energy are, in sum, the addition of such a reagent that the metalliferous particles it is desired to save become coated with an appropriate film. Since this coating is a deposit out of solution, conditions are against its entry into solution; in other words, molecular attraction across the interface between the film and water is low, and, reciprocally, interfacial tension is high. Knowing that, given proper conditions, an electro-negative element is precipitated on one relatively positive, it seems likely that to secure an appropriate film the reagent chosen should contain a metal electro-negative to the metal of the mineral it is desired to float. Electro-chemically the elements have approximately the following order:—Negative end: O, S, N, F, Cl, Br, I, P, As, B, C, H, Pt, Hg, Ag, Cu, Bi, Sn, Pb, Co, Ni, Fe, Zn, Mn, Al, Mg, Ca, Ba, Sr, Na, K: Positive end.

An appropriate film might also result from the adsorption of particles of insoluble soap to the interface. Chemical action between sodium oleate and an appropriate curdling agent would give soap particles of such small size as would readily be adsorbed. Experimentally this method has been successfully tried in the flotation of cassiterite.

To decrease Interfacial Tension.—If, instead of promoting the flotation of the mineral, it be desired more surely to encompass the sinking of the gangue, then a reagent capable of increasing the adhesion with water and thereby lowering the interfacial tension, is added; among others, a reagent capable of chemical attack, incipient or pronounced, upon the gangue would act appropriately; in particular cases sulphuric acid, caustic soda, sodium silicate, etc., are such wetting agents.

Increase in the wetting power of water by the addition of acid is well established, and acid is much used to minimize in this way any tendency of gangue minerals to float, a cleaner concentrate resulting. It has this effect even when it might be supposed that the gas resulting from its action would tend to float the attacked particle; in the Cœur d'Alene district, for instance, the addition of acid sinks unwanted siderite, while galena floats. Acid accordingly decreases the interfacial tension, this decrease being manifested by diminished contact and hysteresis angles. As already mentioned, it also tends to break down excessive emulsion of the oil, permitting that important agent to spread, eventually to pass into the froth, while leaving the pulp beneath apparently deflocculated. Finally, acid will clear away an incipient oxide from a sulphide surface (p. 433).

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[1] Smith and Pickett, *E. & M.J.*, Feb. 22, 1919, p. 365; McLeod, *E. & M.J.*, Aug. 6, 1921, p. 213.

Caustic soda, acting chemically upon the oil, will reduce the interfacial tension of the oil with the water, without at the same time appreciably altering the water tension; the introduction of this agent accordingly increases the spread of the oil and permits the oil to act more quickly. In an alkaline circuit, calcite, not being so likely to be attacked, has a greater tendency to float than in a neutral circuit. Caustic soda also deflocculates colloidal slime, keeping such particles from being adsorbed to the mineral faces, to the exclusion of oil. Deflocculation, which may be regarded as an advanced stage in wetting, follows from the ionization of this reagent, the negative hydroxyl ions completing the dispersion.

Silicic acid has a similar wetting effect. In water this salt hydrolizes, and silicic acid a gelatinous colloid is formed, which being adsorbed upon the solid surfaces lowers the interfacial tension. This colloid also inhibits flocculation, the total effect being a deflocculation so persistent that even the concentrate is rendered difficult to settle.

Effervescence and Froth.

Since the ore submitted to flotation consists of fine particles and these are borne in water, the air surface to which they are to attach themselves must be introduced into the water in a manner to offer as large a total extent and as perfect a distribution as possible: there must indeed be a fine diffusion of minute air bubbles, a mist of air bubbles in water. This distribution is achieved by mechanically dividing the air in the presence of an effervescing agent which both promotes and preserves the fine division (pp. 423, 426).

The effervescing agent is usually a soluble oil or the soluble fraction of a composite oil, though it may be a soluble salt, such as sodium chloride. Molecules of this agent or contaminant being adsorbed to the bubble surface stiffen that surface so that it resists contraction, and bubbles when they meet do not coalesce. Probably, also, the electrostatic charges which these bubbles possess are a factor militating against coalescence, though it must be remarked that in pure water the possible repulsion of these electric charges is not sufficient to prevent coalescence. Coalescence follows in obedience to the tendency of all systems containing potential energy to find equilibrium under conditions where this energy is at a minimum; two bubbles meeting tend to coalesce because the surface of the single bubble containing them both would be less extensive than the sum of the two surfaces. The effervescing agent also opposes coalescence by reducing the surface tension of the water, this opposition being proportional to the extent of that reduction; reduction of surface tension is, however, generally less than 10 per cent.

To these air bubbles once formed, the mineral particles attach themselves, the bubbles being strengthened thereby.

Arrived at the surface these loaded bubbles burst into others of large dimension, which, being out of the water yet consisting largely of water, have not a single air-boundary as have the minute bubbles from which they sprang, but a double boundary with water molecules between; differing further, without the attached mineral particles these froth bubbles would not persist (p. 425).

Concerning bubble-formation generally, surface tension being constant at constant temperature, the slightest change of load resulting from evaporation, drainage between the two surfaces, or additional tension at the top due to the weight of the sides, etc., causes a pure-water bubble to break immediately. With an appropriate contaminant present, conditions are different; adsorption of the contaminant to the two boundaries provides a compensating mechanism whereby change of load is met by change of surface tension, and the bubble is rendered stable; as the double-sided film becomes stretched water molecules are forced from their medial position to the surface to increase its strength, these molecules returning to their medial position should contraction take place.

Stability of the bubble accordingly requires a surface tension capable of automatically varying itself, the degree of the stability being proportional to the range of this tension-variation. Since reduction of surface tension may be brought about by a small quantity of an appropriate contaminant whereas increase requires a relatively great quantity, contaminants which reduce the surface tension, soaps, for instance, are those commonly used. The molecules of these contaminants are adsorbed to the water surface—positive adsorption—where reduction of the tension results from their introduction among the water molecules at that boundary. Contaminants which tend to increase the water tension by being adsorbed away from the surface—negative adsorption—have to be used in greater amount to secure the same variation in tension; sodium chloride, for instance, requires to be present to the extent of 3—4 per cent; the soluble salts which at Broken Hill must be counted to have provided the effervescing and frothing agents were present in similar amount.

In mineral flotation, however, stability of bubble is not all that is required, or the soap bubble would have been the first to have found application; the surface tension of the contaminated water surface must still be sufficient to suspend the mineral, a condition satisfied neither by soap bubbles nor by the bubbles produced by saponine, etc.

Conveniently enough, the mineral particles attached both to the outside and inside surfaces of the bubble-film fill the rôle of an appropriate contaminant perfectly, some of the selective agents, the tars, for instance, assisting. Accordingly, no special purely-frothing agent is used, but the froth remains essentially a mineral froth.

In addition to providing stability, the froth at surface also provides the necessary large extent of surface that the particles may be held without undue clustering; in an ordinary flotation machine acres of water-surface are available in the froth. At the re-entrant angle where separate bubbles meet, there may be clustering and the mineral armour may be measured many particles deep, but in the absence of any depth of water the particles do not fall. Similar clustering below the water surface would be fatal, each particle must be separately held.

Colloidal Electrostatic Phenomena.

The material submitted to flotation consists in preponderating part of particles which in water form a ‘coarse suspension’; the lower limit to the size of these particles may be set at the hypothetical 1200-mesh, or say 10μ. Such particles being apparently subject solely to gravity settle more or less readily. The resulting sediment can be whipped again into suspension; sediment and suspension are thus readily reversible.

In subordinate part it consists of particles still smaller, yet visible by the microscope, that is, of particles from 10μ to $0{\cdot}1\mu$ in mean diameter. With mass decreasing as the cube of diameter, gravity no longer remains the controlling force, but other forces arising from the greatly-increased particle-surface and the inherent kinetic energy of extremely fine particles, assert themselves, as witness the Brownian movement. Such particles in water constitute a ‘fine suspension’; their suspension is prolonged (p. 217).

Finally, in yet smaller part, there will be particles still finer, invisible to the ordinary microscope because their diameter is less than the wave-length of light, but visible to the ultra-microscope by the cone of light which each is capable of reflecting; the lower limit of such particles may be set at $5\mu\mu$. Under the action of inherent energy and the electrostatic charges associated with the relatively enormous surface, the suspension of these particles is continuous. The materials from which they arise are 'colloids,' though, since suspension is so obvious, they likewise are described as 'suspensoids'; the system with water is a 'colloidal solution.' Of such a solution or 'sol,' the particles are the 'disperse phase' and the water the 'continuous phase.' Though from a colloidal solution no settlement takes place, conditions may be so altered by the addition of an electrolyte that, the repellent electric charges being reduced or discharged, the particles assemble in gelatinous floccules which settle readily (p. 258). Such a transformation into floccules is irreversible, the colloidal solution cannot be directly reformed; when the floccules are dried their absorbed water is driven off, they lose their gelatinous character and do not take up water again. The prolonged settlement of a fine suspension may similarly be hastened by the addition of an appropriate electrolyte.

When, instead of solid particles, the parallel occurrences of an immiscible liquid such as oil be considered, then in the place of suspensions and suspensoids come 'emulsions' and 'emulsoids' respectively, the system with water again constituting a colloidal solution; the natural emulsoid colloids, gelatine, glue, etc., were the first colloids. Emulsoid colloids are, with few exceptions, of organic origin, the only important inorganic emulsoid being silicic acid. As a rule, in suspension they are not flocculated by an electrolyte, though to this rule the emulsions of oil constitute exceptions; moreover, this property of non-flocculation they are able to extend to suspensoid colloids in the same suspension, earning thereby the name of 'protective colloids.' Their outstanding characteristic is, however, that under appropriate conditions most of them stiffen to an elastic jelly, a colloid so coagulated being described as a 'gel'; this transformation into the gel condition is generally reversible, a jelly may resolve itself into the watery condition again.

When ordinary gels are dried their absorbed water is gradually driven off, the gel becomes more and more viscous, till, finally, the solid condition obtains. This solidity is not reached at a particular temperature, as when liquids solidify, nor, unlike the combined-water of certain salts, is the water driven off at a particular temperature, it goes gradually. The resultant dried mass will, however, avidly absorb water to return to its original condition and bulk; an exception, again, is silicic acid.

Restating the position, particles suspended in water are not only under the action of gravity but also subject to molecular forces largely consequent upon electrostatic charges they carry; and the balance between these gravitational and molecular forces is such that the larger particles appear to be subject solely to gravity and the finest particles only to the electrostatic forces. These latter, like surface tensions, are a display of surface energy; both are characteristically colloidal phenomena, in that both assume a greater importance when the particles are of colloidal size. Unlike surface tension, however, which for its display requires at least three phases to be present, say, particle, water, and air, the electrostatic forces manifest themselves also in a simple aqueous suspension, without air.

In such a suspension solid particles, mineral and gangue alike, mostly possess a negative charge ; in an electric field they move towards the anode or positive electrode. So charged they repel one another, this repulsion being sufficient to prevent the settlement of colloidal particles. An electrolyte added may either reinforce these charges, reduce them, or reverse them. If the electrolyte be the salt of a metal electro-negative to the metal of the mineral, and particularly if the mineral itself be a conductor, the positive metallic ions of the electrolyte, that is, the kations, are adsorbed to the surface of the mineral, reducing the negative charge of the particle and eventually reversing it. Sulphuric acid appears in this respect to act as such an electrolyte, the hydrogen ions being the adsorbed kations. Speaking generally, ordinary metallic minerals, and particularly the sulphides, are conductors, while the gangue-minerals are non-conductors. The most striking result of the addition of such electrolytes in appropriate amount is the flocculation of the dispersed particles, this flocculation following upon the discharge, partial or complete, of the original negative charges. With the electrolyte insufficient in amount to effect the flocculation of both mineral and gangue, the mineral would be selectively flocculated (p. 258).[1]

On the other hand, if the electrolyte be an alkali or an alkaline salt which ionizes in solution, the kations are inappropriate for adsorption to the mineral surface, since the alkalies are electro-positive to all other metals ; under these circumstances the negative hydroxyl or acid ions, that is, the anions, stimulate greater repulsion of the suspended particles, with the result that deflocculation becomes pronounced.

It may here be remarked that though they permit current to pass, electrolytes do not conduct away the charges on suspended particles ; it is through the free ions that the current passes. It will also be recalled that the minute particles of an oil emulsion behave like suspensoids rather than emulsoids, that is to say, they suffer flocculation and deflocculation similarly to suspensoids.

From the foregoing it would appear that in a simple suspension some degree of selective flocculation is possible, though, at best, no great step in enrichment can be conceived, since in the subsequent precipitation much of the gangue would be mechanically involved.

In flotation, however, the suspension is something more than one of ore particles, it includes a swarm of air bubbles. Though these bubbles are large enough to float to the surface under the control of gravity, their formation and movement are also subject to molecular forces. Ordinarily, they are negatively charged; in alkaline suspension, on the other hand, they appear to possess a slight positive charge.[1] Assuming a negative charge, it is conceivable that, between the air bubble and the mineral particle which had been rendered positive, there might be an attraction capable of promoting the attachment of colloidal particles to air bubbles; to the positively-charged air bubbles of an alkaline suspension the negatively-charged mineral particles would also, conceivably, proceed, crowding out the gangue particles by their heavier charges (p. 501). However satisfactory such an explanation based on electrostatic phenomena may appear in respect to suspensoids, there seems little doubt that interfacial tensions, which operate upon suspensions and suspensoids alike, remain the essential basis of flotation.

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[1] Parsons, *M. & S.P.*, Nov. 6, 1920, p. 661.

That electrostatic charges contribute to the functioning of the process is, however, certain from the profound effect which the smallest amount of electrolyte, of colloid, or of gas, can have upon the result. The addition of acid has sometimes and almost miraculously secured a mineral froth when neutral and alkaline circuits have both failed; a trace of a negative colloid, such as glue, saponine, tannine, etc., has sometimes hopelessly destroyed all tendency of the mineral to rise; a whiff of sulphur dioxide in the air has sometimes caused the froth to collapse and disappear.

In conclusion, the prime factor in flotation, namely, the attachment of air to mineral, is the triumph of molecular forces, surface tension in greater measure than electrostatic attraction, over gravitational forces. These molecular forces being capricious and changing with apparently trivial change of circumstance, the process needs close observation and great manipulative skill to secure the best results; such necessary attentions do not require a host of men, but are rendered almost in advance by the experienced and properly trained worker.

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[1] Corliss & Perkins, *Jour. Ind. & Eng. Chem.*, May 1917. Abst. *M. & S.P.*, June 9, 1917, p. 809.

CHAPTER XI

MAGNETIC SEPARATION

GENERAL

UP to the 'nineties' of last century, miners recognized only two minerals as naturally magnetic, namely, magnetite and pyrrhotite. In the knowledge, however, that the non-magnetic iron-minerals could be made magnetic by roasting, magnetic separation, as practised, was founded on both the natural and the artificial magnetic properties; for instance, crude magnetite was concentrated from its ore, and, after roasting, siderite and pyrite were removed from zinc concentrate.

It was not till Wetherill in 1895 at Franklin, New Jersey, by the use of pointed poles, employed the greater attractive strength of diverging fields, that the wider possibilities of magnetic separation became manifest, and feebly-magnetic minerals were counted as coming within its scope.

Magnetic separation becomes possible by the distinctive movement of magnetic minerals towards a magnet, non-magnetic minerals passing on. This movement is the response to an attractive force of magnetism. The space around the magnet in which this force is exerted is the 'magnetic field.' Ordinarily this field is in atmospheric air. It is visualized as lines of force diverging from the north pole of the magnet, eventually to converge upon the south pole, the inner lines after a relatively short circuit through the air, the outer lines after a circuit of infinite length; at each pole, neglecting the leakage at the sides, the magnetic mass to which the field is due is concentrated, the strength of the field diminishing with the square of the distance from that mass (Fig. 351). Within the magnet the lines of force, running from the south pole to the north and thus completing the circuit, constitute the 'induction.'

Magnetic lines of force being capable of passing through all substances, all substances are said to be permeable to magnetism. Passage, however, is not equally ready through all substances, some being more permeable than others. The facility with which a substance passes magnetic lines of force, expressed in relation to passage through air as unity, is described as the "magnetic permeability" of the particular substance.

According to the permeability of their substance, particles brought within a magnetic field may either concentrate the lines of force within themselves, becoming polarized and consequently attracted; or, the lines of force may pass preferably through the air, with consequent repulsion. Particles capable of concentrating the lines of force align themselves with those lines, wherefore the substance of which they consist is said to be "paramagnetic." Particles which disperse the lines of force align themselves at right angles to those lines, their substance being "diamagnetic."

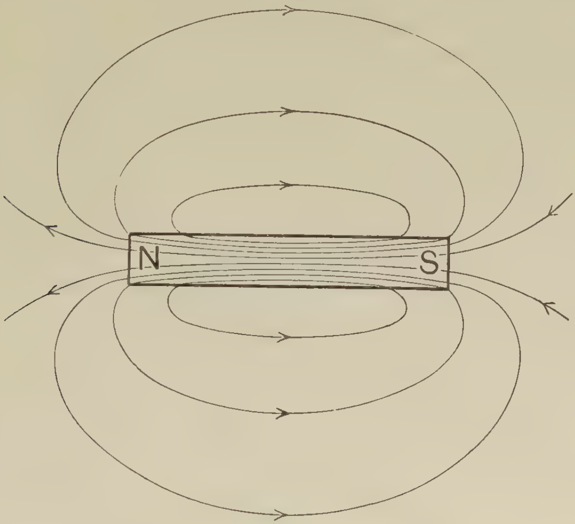

FIG. 351.

Magnetic Circuit around a Bar Magnet.—Diagram. The lines of force outside the magnet constitute the field, $\mathcal{H}$; the lines of force within constitute the induction, $\mathcal{B}$. The outer lines of the induction, not continuing to the poles, are shown leaking from the sides (p. 514).

The substances defined above as paramagnetic are those ordinarily described as magnetic, that is to say, are those attracted by a magnet. The phenomenon of magnetic attraction is common, particularly in the field of the electro-magnet; often indeed it is pronounced and forceful, the permeability of iron being, for instance, about 2000 times that of air. The phenomenon of repulsion, on the other hand, is rare, the repulsive force being too small to take effect; the permeability of bismuth, the most diamagnetic metal, is, for instance, 0·998, or practically the same as air.

Accordingly, to ordinary intents and purposes diamagnetic substances are non-magnetic.

The investigations of Faraday, Plücker, and others, have established that among the elements, those of the iron group, namely, iron, nickel, cobalt, chromium, etc., and the uncommon metals, cerium, palladium, osmium, platinum, etc., are paramagnetic; while bismuth, sulphur, copper, antimony, zinc, mercury, lead, silver, tin, etc., and all the gaseous elements except oxygen, are diamagnetic.

Compounds and alloys, though in general not belying the magnetic properties of their constituent elements, have their own. Oxides quite normally are more paramagnetic than sulphides; the compounds of iron are as a rule pronouncedly paramagnetic. But bromide of copper, compounded of two diamagnetic elements, is paramagnetic; equally abnormally, an alloy of manganese, aluminium, and copper, all individually non-magnetic, is magnetic.

It is still less safe to predict the magnetic properties of a mineral from consideration of the elements normally constituting the mineral. Those magnetic properties may depend upon impurities present, or upon the isomorphous replacement of one element by another. Pure blende is, for instance, non-magnetic, but when iron replaces zinc to the extent that 12—14 per cent of iron is contained, the mineral, then known as marmatite, shows itself magnetic even to the ordinary magnet.

Even the iron content of minerals, though this largely determines the magnetic properties, is no invariable guide to those properties; witness the non-magnetic character of ordinary pyrite, a mineral almost half constituted of iron. Pyrrhotite, on the other hand, though compounded of the same two elements iron and sulphur, is strongly magnetic; in this mineral the iron exceeds the sulphur.

Accordingly, only by actual test is the magnetic character of any particular mineral determined reliably.[1] Thus determined, minerals are divided into three classes: firstly, those which are so strongly magnetic as to be attracted by an ordinary magnet, these being known as "ferro-magnetic" minerals; secondly, those which, while not responding to the ordinary magnet, may be made to move by electro-magnets, these being the "feebly-magnetic" minerals; and thirdly, the class of "non-magnetic" minerals.

Ferro-magnetic are: magnetite, FeO, Fe_2O_3; pyrrhotite, Fe_7S_8; ilmenite, FeO, TiO_2; chromite, FeO, Cr_2O_3; franklinite $(Fe, Zn, Mn)O, (FeMn)_2O_3$; and marmatite $(3Zn, Fe)S$, in representative specimens.

Feebly-magnetic are: siderite, haematite, limonite, garnet, wolframite, monazite, rhodonite, etc.

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[1] W. R. Crane, *Trans. A.I.M.E.* Vol. XXI., 1902, p. 444.

Some idea of the relative susceptibilities of the more prominent of these minerals to magnetization may be gathered from the following scale, wherein for comparison the magnetic susceptibility of iron is placed at 1,000,000 :

Soft iron	1,000,000
Magnetite	40,000
Siderite	800
Haematite	400

THEORETICAL CONSIDERATIONS

Susceptibility, Permeability, and Magnetization.—Magnetic susceptibility is the ratio of the intensity of magnetization to that of the field creating it ; the susceptibility of air is zero. Using the symbols of the physicist :

$$k, \text{ the magnetic susceptibility} = \frac{\mathfrak{I}}{\mathfrak{H}}.$$

Magnetic permeability is the ratio of the induction to the magnetizing field, that is to say, it is the ratio of the internal field to the external field producing it ; the permeability of air is 1.

$$u, \text{ the permeability} = \frac{\mathfrak{B}}{\mathfrak{H}}.$$

The relation between susceptibility and permeability is obtainable from the formula connecting induction, field, and magnetization, namely, $\mathfrak{B} = \mathfrak{H} + 4\pi\mathfrak{I}$. Dividing by $\mathfrak{H}$

$$\frac{\mathfrak{B}}{\mathfrak{H}} = 1 + 4\pi\frac{\mathfrak{I}}{\mathfrak{H}}$$

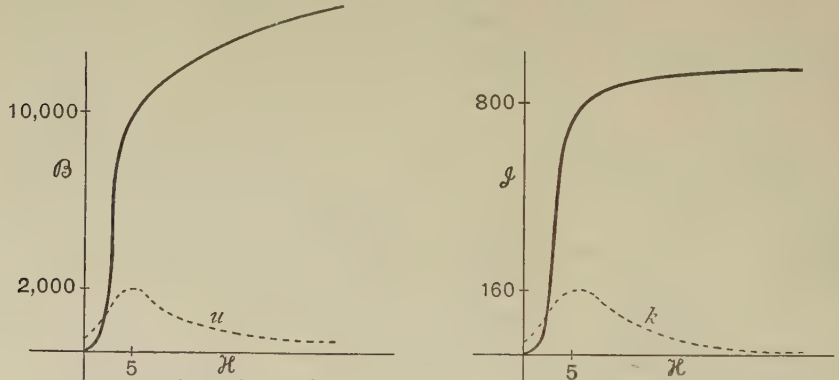
or,

$$u = 1 + 4\pi k \text{ ; and } k = \frac{1}{4\pi}(u-1).$$

In conformity with this relation, ratios of magnetic susceptibility have a wider range than those of permeability. At its maximum the permeability of soft iron is as much as 2000 ; that of most other substances is not far removed from unity, of paramagnetic substances a fraction greater and of diamagnetic the minutest fraction less than unity. The corresponding extremes for susceptibility are 160 and the minutest fraction less than zero. Susceptibility is a function of the difference between the induction and the magnetizing field, that is to say, it is a function of the magnetization remaining after the strength of the field has been deducted.

$$\mathfrak{I} = \frac{\mathfrak{B} - \mathfrak{H}}{4\pi}; \quad k = \frac{\mathfrak{I}}{\mathfrak{H}} = \frac{\mathfrak{B} - \mathfrak{H}}{4\pi\mathfrak{H}}.$$

Neither the permeability nor the susceptibility of a substance is, however, constant, since neither the induction on which one depends nor the magnetization on which the other depends, increases directly with the magnetizing field. With most substances, since both permeability and susceptibility are low, the variation is not great. But with highly permeable iron or steel it is extreme. Starting with a weak field both the permeability and the susceptibility of soft iron are low. At a field strength of about 5 lines of force per square centimetre the induction has climbed abruptly to about 10,000 and the permeability consequently to 2000, roughly its maximum value (Fig. 352). From that point, with increase in field strength the induction rises more slowly, and the permeability falls, steeply at first and gradually afterwards, till with a field of about 16,000 lines of force it has fallen to about 2. It is clear that in a field of infinite strength the permeability of iron and of all things would be unity.



Induction and Permeability Curves

Magnetization and Susceptibility Curves

Fig. 352.

Permeability and Susceptibility Curves.—The curves drawn are roughly those for soft iron or mild steel (p. 518).

Similarly, the graph of susceptibility rises abruptly to a maximum in a field of about 5, whence it descends steeply at first and gradually afterwards, till in an infinite field it would be zero. Long before this, however, increase in magnetization is at so slow a rate that practically speaking the iron has become 'saturated,' a condition in which the molecular magnets of which the iron may be taken to consist, would all be oriented along the lines of force and nothing remained to be accomplished. In magnetic separation, saturation is generally considered to be reached when the induction is of the order of about 16,000 lines of force per square centimetre; in dynamic electric machines, motors and generators, a lower figure, some 8000—10,000 lines, is considered to bring about economic saturation.

The degree of magnetization which connotes saturation falls as the temperature rises; with iron or steel this fall continues gradually and regularly till a temperature of about 785° C. is reached, when it is critically abrupt and all magnetization ceases. At that temperature, described by metallurgists as the temperature of 'recalescence,' there is, demonstrably, molecular rearrangement.

From the foregoing it will be realized that the permeability of a substance, while peculiar to that substance under given conditions, varies with the field and temperature. With field and temperature constant, determinations of permeability may be rendered inconstant by other varying factors; crushed material, for instance, will by reason of interstitial air give different figures to massive material.

Though, therefore, it is convenient to attribute differences in magnetic behaviour to specific permeabilities, and though the order of permeability may be established, no precise figures for that property are of general application; the possibilities of magnetic separation in any particular instance must await determination by actual test.

Electro-magnetic Circuits.—The induction of an ordinary permanent magnet is about 800, this figure being obtainable from the measured lifting force of magnets, by the application of the formula

$$F, \textit{ the lifting force in dynes} = \frac{\mathfrak{B}^2 S}{8\pi},$$

where S is the cross section of the magnet and $\mathfrak{B}$ the induction, that is, the number of lines of force per square centimetre.

Such a low induction is impotent in modern magnetic separation, where both the nature of the separations effected and the considerations of economy demand the higher figures readily obtainable by the electro-magnet. The electro-magnet is an iron or steel core upon which an insulated conducting wire is closely wound, layer upon layer. The length of this core is generally substantially greater than its transverse dimension, so that the wire-envelope approximates the geometric figure of a solenoid.

The field of a solenoid is given by the formula

$$\mathfrak{H}, \textit{ the solenoid field} = 4\pi n I \ 10^{-1},$$

where n is the number of wire-turns per centimetre of the core length and I the number of ampères of current passing.

The field of a flat or ring coil, with transverse dimension great relatively to its length, is proportional to the total number of turns, and inversely proportional to the coil radius, thus

$$\mathfrak{H}, \textit{ the ring field} = \frac{2\pi N I \ 10^{-1}}{r}.$$

Accordingly, since $N/2r$, the ratio between these two fields, would invariably be less than unity, the field of a ring coil is substantially less than that of a solenoid coil.

Applying the formula for the solenoid to a magnet wound 250 turns per centimetre, then for each ampère of current passing, the field would be $1{\cdot}25 \times 250 = 312$. In such a field the permeability would be about 70 for a perfect iron-circuit such as would be formed by a complete annular core, and say about 35 for a straight core necessitating a return circuit through the air. With this latter design the induction would accordingly be of the order of 10,000. Employing 500 turns and 10 ampères, the field would be $1{\cdot}25 \times 500 \times 4 = 2500$. In a field of that strength the permeability of a perfect iron-circuit would be say 12, and that of a straight core about 6, at which latter figure the induction would be about 15,000.

The foregoing considerations illustrate the effect of the nature of the circuit upon the permeability of the circuit, and consequently upon the induction. The specific permeability of a substance is that obtaining when the circuit is homogeneously and completely of the particular substance. The average permeability of a heterogeneous circuit, say of iron and air, is calculable by 'weighting' the permeabilities of the different media with the lengths of the traverses through those media, respectively.

Approaching this subject from the circuit instead of from the field, the force compelling the circuit, that is, the 'magneto-motive force,' is proportional to the total number of turns and also to the intensity of the current passing; thus with the current expressed in ampères

$$\mathfrak{F}, \textit{ the magneto-motive force} = 4\pi NI\,10^{-1}.$$

The force resisting the circuit, the 'reluctance,' is directly proportional to the length of the circuit, inversely proportional to the section, and inversely proportional to the average permeability of the circuit.

$$\mathfrak{R}, \textit{ the reluctance} = l/uS.$$

Finally, the magnetic flux around the circuit is the quotient of the magneto-motive force divided by the reluctance,

$$\Phi, \text{ the flux} = \frac{\mathfrak{F}}{\mathfrak{R}}, \frac{\textit{Magneto-motive force}}{\textit{Reluctance}}.$$

But flux is the product of induction multiplied by section, or $\mathfrak{B}S$; therefore

$$\mathfrak{B}S = \frac{4\pi NI\,10^{-1}}{\dfrac{l}{uS}},$$

whence $$\mathfrak{B} = u4\pi\frac{N}{l}I\,10^{-1}; \text{ or, } u4\pi nI\,10^{-1}.$$

That is to say $\mathcal{B}$, the induction, is equal to the product of the magnetizing field multiplied by the average permeability of the circuit.

$$\mathcal{B} = u\mathcal{H}, \text{ or } u = \frac{\mathcal{B}}{\mathcal{H}}.$$

Accordingly, the induction $\mathcal{B}$ may be increased either by increasing the permeability of the circuit, or by increasing the strength of the exciting field, and doubly by increasing both.

It is obvious, therefore, that the induction is largely dependent upon the character of the exterior circuit. In magnetic separation there are broadly three dispositions of this exterior circuit: The first, where a single pole radiates its lines of force into space, these eventually returning to the pole of opposite polarity some distance away; such a disposition is only rational with ferro-magnetic materials, where high induction is not required (Fig. 353). The second, where the exterior circuit is of iron except for an air-crossing or 'gap' at either pole; such a disposition secures an induction strong enough to attract feebly-magnetic materials. And thirdly, where the exterior circuit is through another primary magnet wound to an opposite polarity; such a disposition, by confining the lines of force for much of the exterior circuit, prevents leakage at the sides and secures maximum induction at the poles.

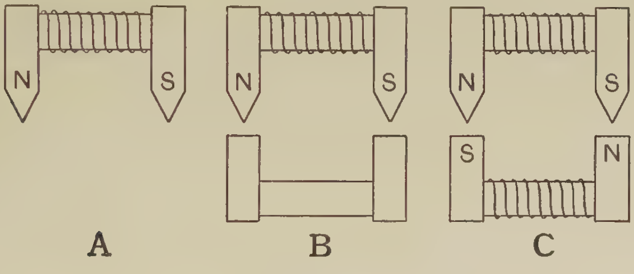


FIG. 353.

Representative Circuits in Magnetic Separation.—Diagram. A represents the circuit largely completed in air, when there will be considerable dispersal of the lines of force. B represents the circuit completed by an iron bridge which concentrates the lines of force and diminishes leakage. C represents the circuit when the place of the iron bridge is taken by a second electro-magnet, the windings of which confine the lines of force. The permeability of the heterogeneous iron-air circuit of A is considerably less than that of B, where consequently the induction and the field at the air-gap are stronger. In C, leakage of lines of force through the windings of the second magnet being impossible, the induction in that circuit and the field in the air-gaps are stronger than in B, and stronger still than in A (p. 521).

Given a winding long enough to approach the solenoid figure, extra length, though it would create greater magneto-motive force, would create a proportionally greater reluctance; flux and induction accordingly would not be increased. But, ordinarily, the winding falls short of solenoid length, and the field strength is less than the solenoid would give; where, then, greater induction is required, the length of winding is increased with advantage. Similarly, extra layers upon the winding, though they increase the magneto-motive force and consequently the induction, do so at the low rate of a ring coil; moreover, on account of their greater diameter, such outside layers take greater length of wire for the same number of turns. Nevertheless, when great induction is required the winding must be relatively deep.

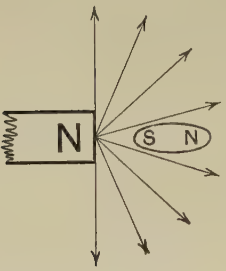


Fig. 354.

Attraction of a Particle in a Diverging Field.—Diagram. Under the influence of the magnet the particle becomes polarized, developing an attractive pole at its near end where the field is stronger, and a repulsive pole at its far end where the field is weaker. In a radiating field there is uniform and regular divergence; but any divergence, however small and irregular, would result in attraction (p. 522).

Ordinarily the length of winding varies from 10 to 30 in., and the depth from 2 to 4 in. With respect to the size of wire, experience shows that with insulated wire not more than 2·5 ampères can be carried per square millimetre of copper section, without risk of undue heating (p. 550).

The Force of Attraction.—Magnetic attraction is a display of the force which in any system possessing potential energy works towards the position of equilibrium where that energy is least. In response to that force a magnetized particle in a magnetic field moves in the direction of greater field strength, the system thereby doing work against any outside reaction tending to stop that movement. Particle attraction connotes movement in the direction in which the lines of force converge; repulsion, movement in the opposite direction of divergence (Fig. 354).

Where no variation of the field strength exists, that is, where the field being uniform the lines of force are parallel, though the particle may become strongly magnetized it will not move except to align itself with the lines of force; the terrestrial magnetism at any one place is an example of such a field (Fig. 355). This non-movement of a particle results from the equal and opposite forces acting upon the two poles. A single pole would move even in a uniform field.

The force attracting particles is in fact proportional to the rate of change of the field strength in respect to distance; it is, in addition, also proportional to the magnetic susceptibility of the particle and to the volume of the particle, the whole position being represented by the expression

$$F = KV\frac{d\mathscr{B}^2}{ds}.$$

In certain cases all the terms of this expression are known. In a normally radiating field the strength of the field varies inversely as the square of the distance from the pole, and the force acting between two masses of magnetism m and m', with a distance r between them, is given by Coulomb's Law—

$$F = \frac{mm'}{r^2},\ \text{or}\ m\frac{m'}{r^2}.$$

This law may be applied to the determination of the lifting force of a magnet of a given induction $\mathscr{B}$ and section S. From considerations of the rate of change of the field strength as the pole is approached, namely, $1/r^2$, the strength of the field at a point situated very close to the pole is $2\pi\sigma$, where σ is the density of the magnetic mass. But since $\mathscr{B}$ is equal to $4\pi\sigma$, σ is equal to $\mathscr{B}/4\pi$, this density existing both on the magnet and on the face of the iron lifted; the two magnetic masses are accordingly each equal to $S\mathscr{B}/4\pi$. The force of attraction between them is the product of one mass multiplied by the normally radiating field of the other, the magnetic masses on the distant poles being too far away to interfere.

$$F = \frac{S\mathscr{B}}{4\pi} \times 2\pi\frac{\mathscr{B}}{4\pi} = \frac{S\mathscr{B}^2}{8\pi}.$$

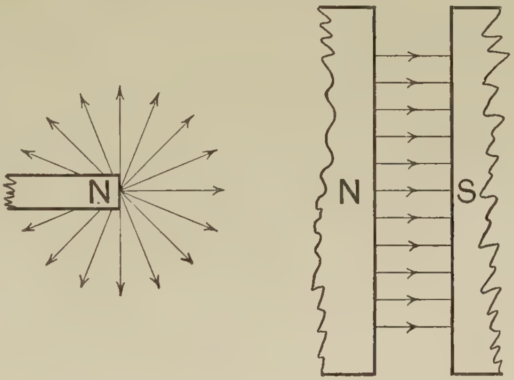


Fig. 355.

Radiating and Uniform Fields.—Seeing that the total flux crossing a closed surface containing a pole is constant whatever the distance from the pole, it is easy from considerations of a spherical surface to deduce that in a radiating field the strength of the field varies inversely as the square of the distance from the pole. A particle in such a field would have its nearer pole more strongly attracted than the farther pole was repelled, and consequently would move. In a uniform field the lines of force have all the same direction and intensity, and accordingly in such a field a polarized particle does not move, one pole is as much attracted as the other is repelled; a single pole would, of course, move (p. 522).

A paramagnetic particle placed between two plane poles not far apart is, however, in a different position, and no possibilities of useful calculation exist. Such a particle is completely enveloped in a field where the divergence of the lines of force, that is, the rate of change of field strength with distance, is small; indeed, across a medial plane the rate of change is *nil* and the lines of force are parallel; the field is uniform (Fig. 356). Further, upon the particle two equal and opposite masses of magnetism are induced, both of which are in the field, the one being attracted, the other repelled; in consequence, any remaining force of attraction is the excess over a nearly equal repulsion. A particle precisely balanced across the medial plane where the diverging lines of force from one plane change their direction to converge upon the other, would experience no attraction however strong the field and however completely the particle were magnetized. On either side of that plane, however, the particle would move in the direction of the converging lines of force, finally coming into contact with the pole.

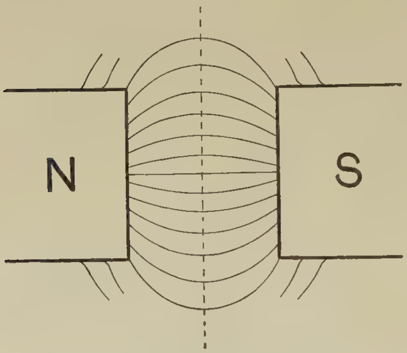


FIG. 356.

Uniform Field medially between Two Plane Poles.—Diagram. As the lines of force diverging from one pole change direction to converge upon an equal and opposite pole, they are momentarily parallel to one another; that being so, the field crossing the medial plane between two poles, theoretically speaking, is uniform (p. 524).

From the foregoing considerations it is clear that though a strong induction makes for attraction by more strongly magnetizing the particle, it is ineffective unless at the same time the field changes strength with distance, the particle will not move except towards greater field-strength. The greater attraction of diverging lines of force is seen in the bunching of iron filings upon the edges of a magnet.

Accordingly, in magnetic separation plane poles are not set facing one another but are used alone or facing wedge-shaped poles; the latter disposition gives the greatest rate of change of the field strength, since the lines of force rapidly converge from the plane to the wedge pole, or rapidly diverge if regarded as proceeding in the opposite direction (Fig. 357). Wedge poles also give the greatest field strength since they concentrate the lines of force, in the one case those coming from the iron, and in the other those coming across the air; for this reason poles of wedge shape are used surgically, to withdraw iron splinters which have penetrated the eye or other delicate member. In magnetic separation wedge-shaped poles have also been used facing one another, the material being presented to one side and not actually between the poles (Fig. 369).

Time apparently does not enter the question of magnetization, at least that would be the inference from the fact that in alternating current, with alternations as high as 600,000 per second in laminated iron, the limit to the speed of magnetization is not reached. Since, however, the attraction for the mineral particles must be strong enough to cause them to leave the path in which they are travelling, the speed of travel through the field must be adjusted to the attractive force. In this connection particular experiments showed that, of a mixture containing magnetite, rhodonite, and blende, when using a current of say 5 ampères, only magnetite was withdrawn at a speed of 100 metres per minute, whereas at 50 metres the rhodonite was removed, and finally, at 30 metres, the blende. On the other hand, when at a standstill all these minerals were removed by a current of say 0·166 ampères.

In respect to particle size, since the attractive force in a given field would appear to be proportionate to particle volume, that is, to mass, and the forces capable of resisting attraction, namely, gravity, friction, momentum, vary in the same proportion, the field and circumstance suitable for a large particle might also appear suitable for a small particle. It is the experience, however, that while a large particle will readily free itself from a bed of particles and fly upward to the magnet, a smaller particle at the bottom of the bed may not get through; this experience may be explicable by the weight of overburden pressing on the small particle. But the observed fact that when movement is necessary to reach the magnet large particles move before the smaller, is explicable by the greater relative nearness of the attractive end of the particle and the greater relative distance of its repulsive end; the greater the ratio of the particle dimension to the distance from the pole, or, equally, to the width of the air gap, the greater the force of attraction.

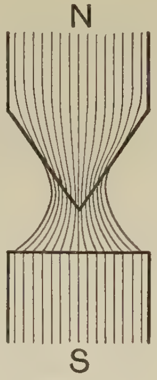


FIG. 357.

Diverging Field at a Wedge Pole.—Diagram. A wedge or pointed pole not only concentrates the lines of force but converges them upon itself, so that particles are attracted to it (p. 524).

When, however, the particles are fed to come in contact with a naked magnet, a revolving magnetic cylinder, for instance, the position is different, the smaller particle is attracted as strongly in relation to its mass as the large particle; indeed in the turmoil of passage the small particle, clinging more closely than the large particle, is more securely held. In such circumstances, experience shows that the larger particle requires a stronger magnet, that is, greater intensity of current.

SCOPE AND PREPARATION

Concentration and Separation.—With magnetite, the most permeable of minerals, the crude ore may be treated magnetically to separate the mineral from the gangue; such a treatment, which may be described as 'magnetic concentration,' is commonly undertaken in the Mineville district of New York, in New Jersey, in Scandinavia, and elsewhere. Similarly, worthless pyrrhotite may be withdrawn magnetically, leaving a product enriched in valuable non-magnetic sulphides; such a treatment was devised for the ore at the Sullivan Mine, British Columbia, though afterwards abandoned in favour of flotation. Finally, a unique magnetic concentration of crude ore takes place at Franklin, New Jersey, where franklinite is magnetically separated from zincite, willemite, etc., and gangue. In this last instance both the magnetic and the non-magnetic products are valuable, the latter being further concentrated by water.

Of all other minerals the permeability is so low that magnetic concentration from the crude ore is economically impossible. With these, magnetic separation becomes limited to the treatment of a concentrate obtained by water; such a treatment may be described as 'magnetic separation' in its narrow sense. So described, common magnetic separations are: blende from roasted pyrite, marcasite, etc., this being widely practised, and particularly in the Mississippi Valley; siderite from blende, this being practised on the Continent, Sardinia, etc.; calcined siderite from gangue on the Continent; wolframite from cassiterite, in Cornwall, Bolivia, Australia; monazite from ilmenite and zircon, in India, Ceylon, and Brazil. Exceptionally, magnetic blende is separated from pyrite, at Leadville, Colorado; smithsonite, zinc carbonate, from limonite; magnetite from bornite, chalcopyrite, etc.

Preparation for Magnetic Treatment.—The first essential to separation is that the magnetic mineral should be released from attachment to non-magnetic mineral, generally by crushing. Where the material treated is the concentrate from a previous treatment by water, that release will already have been effected. Where, however, the concentration of the crude ore is achieved magnetically, release by crushing will be a necessary preparation for that treatment itself; in this respect magnetic concentration does not differ from either water concentration or flotation.

The extent to which crushing is carried depends upon the distribution of the magnetic mineral in the ore, and the grain of the mineral; where the aggregation is coarse, concentration will begin earlier and less material will be submitted to fine grinding. On the Eastern Mesabi, Minnesota, concentration is planned to begin after the ore has been broken to 2 in.

and to end after ball-mill grinding to 100 mesh. At Mineville, New York, and in New Jersey, concentration begins variously from 2 in. to ¼ in. and ends after crushing to 20 mesh, further comminution not being required (Fig. 383). At Herrang, Sweden, concentration does not begin till the ore has been reduced to 2 mm.; at Sydvaranger, Norway, till comminution has been completed in ball-mills to 20 mesh; while at Pitkaranta, Finland, sliming is necessary. These are all magnetites. At Franklin, New Jersey, on the other hand, franklinite is not released and magnetic separation does not begin till the ore has been reduced to 60 mesh.

In general, taking advantage of the coarseness of the magnetite and waste, stage crushing is a feature of the magnetic dressing plants treating the North American magnetites; associated with stage concentration, the finer rolls are relieved of much work, both by the reduction in quantity of material and by the elimination of particularly hard rock. On the other hand, where, as in Scandinavia, all the ore must be reduced to a fine size, dry crushing and separation are excluded because much dust is not only harmful to fast-running machinery but also contaminates the resultant concentrate.

Next to comminution, proper sizing is essential to successful magnetic separation, not only because of its assistance in stage crushing but also because magnetic separators do not function satisfactorily upon material possessing a wide range of size. In general, the lower the permeability of the magnetic mineral, and the smaller the difference in permeability between two magnetic minerals to be separated, the closer the necessary sizing. Even, however, with magnetite, close sizing often secures greater efficiency of the whole plant, particularly where the quantity treated is large enough to add its sanction to this differentiation.

At Mineville, New York, the following sizes are made and treated separately: 2—¾ in.; ¾—⅜ in.; ⅜—¼ in.; ¼ in.—20 mesh, and 20 mesh—0. At the Replogle Mine, New Jersey, there are five sizes, ⅝—3/16 in.; 3/16—5/32 in.; 5/32 in.—6 mesh; 6 mesh—8 mesh; and 8 mesh—0. At Moose Mountain, Ontario, on the other hand, the ore after being broken to pass a 1⅜ in. aperture, was treated unsized, though admittedly at the expense of the recovery of fine magnetite.

In Sardinia, separating siderite from blende the water concentrate is divided into three sizes, namely, 7—4 mm.; 4—2 mm.; and 2 mm.—0, each size being treated on its own separator. At the Launceston Works, Tasmania, separating wolframite from cassiterite and bismuthinite, four sizes are made from 1/8 in. downward. In Cornwall, as a rule, coarse and fine concentrate are treated separately, the general fineness of the cassiterite not warranting any greater subdivision.

Only where the material is ground fine and the magnetic mineral is pronouncedly permeable may treatment take place without sizing, yet with satisfactory recovery. At Franklin, for instance, the ore is ground to 60 mesh and treated in that one size. In Sweden and Finland magnetite ores are ground fine and treated wet without sizing. In general with magnetite, material finer than about 12 mesh requires no sizing.

Finally, an adequate preparation generally includes the drying of at least a proportion of the material treated; and often includes a magnetizing roast.

The drying of crude magnetite ore in preparation for magnetic concentration is generally limited to the smalls which after preliminary breaking are separated by, say, a 1½ in. screen; in that portion of the ore the moisture is usually concentrated, the small amount remaining with the larger material being insufficient to affect freedom of movement among the pieces. Being necessary for the finer dry-separation, such drying is undertaken even if the finest separation is effected in a wet separator and after wet grinding; it finds no place, however, where all the separation is wet. In the concentration of zinc ores at Franklin Furnace, New Jersey, all the ore after being crushed to ½ in. is dried, and then crushed to 60 mesh for separation.

Magnetic separation applied to a water concentrate demands that all the material be dry. This dry condition may be reached by special drying, or it is achieved concomitantly with such roasting as may be necessary to develop magnetism.

In magnetic separation roasting is undertaken to convert non-magnetic or feebly-magnetic iron minerals into ferro-magnetic iron compounds. By a slight roast, pyrite, FeS_2, which is non-magnetic, may be converted into a ferro-magnetic sulphide, Fe_7S_8, and by a somewhat longer roast to the magnetic oxide, FeO, Fe_2O_3; but over-roasted in the presence of abundant air, the ferrous oxide combines with more oxygen till the whole consists of ferric oxide, Fe_2O_3, a feebly-magnetic substance. The iron of chalcopyrite, Cu_2S, Fe_2S_3, bornite, $3Cu_2S, Fe_2S_3$, mispickel, $FeAsS$, etc., undergoes similar changes to become magnetic. Similarly, siderite, the natural iron carbonate, $FeCO_3$, under the action of heat disengages carbonic acid, a short roast in a closed furnace tending to the production of the magnetic oxide, while a longer roast with abundant air again produces the feebly-magnetic ferric oxide.

Where much of the ore is crushed dry to a relatively fine size the preparation for magnetic separation often includes the separation of the impalpable dust by an 'exhauster.' The dust arising in roasting is also sometimes similarly drawn away.

The operations of drying, roasting, and subsequent cooling, are described in a later chapter.

MAGNETIC SEPARATORS

The machines which accomplish magnetic separation must provide the necessary magnetic field; must present a proper stream of ore into and through the field; and must remove the magnetic minerals, the non-magnetic continuing with the stream. If that stream be dry, such machines are 'dry separators,' whereas if the material be water-borne and in the condition of a pulp they are 'wet separators'; mostly, magnetic separators are dry, wet separators being only practicable with finely-crushed ferro-magnetic material.

Separators may also be described as drum, pulley, belt, ring, disc, armature, and motor separators respectively, after some prominent feature in the design.

Finally, though no difference in principle exists, it is convenient to differentiate between Ferro-magnetic and Feebly-magnetic separators.

Ferro-magnetic Separators.

The simplest magnetic separators are those particularly applied to the recovery of magnetite, namely:

Ball-Norton Drum Separator.—This separator, used largely in North America, consists of a drum of non-magnetic metal, manganese-steel, or rubber-covered brass, 24—36 in. diameter and 18—28 in. face, making 50—60 revolutions per minute around a fixed vertical yoke occupying roughly one vertical half of the circle. In that yoke are held about a dozen radial magnets so wound that opposite poles follow one another close under the inner surface of the drum. Such alternate polarity, in continually breaking and remaking the magnetic loops, gives opportunity for entangled waste to clear itself; moreover, the juxtaposition of opposite poles strengthens the field. On the top of the drum the material to be treated is delivered in the direction of rotation and over the magnets. Carried forward, the non-magnetic material drops straight

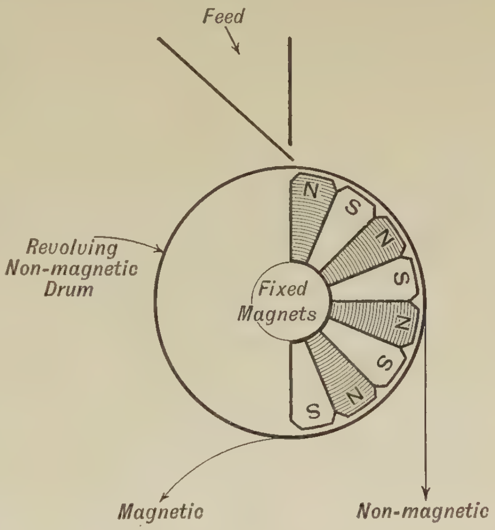


FIG. 358.

Ball-Norton Drum Separator.—Diagram. For ferro-magnetic and coarse material, coarse magnetite for instance (p. 529).

off, while the magnetic material is held against gravity and centrifugal force till the magnets are passed, when it is flung off (Fig. 358).

With 4—5 ampères of current and treating per hour 20—30 tons of ore crushed to 2 in., this machine separates clean magnetite in what is described as a cobbing operation, the bulk of the material remaining to be further treated. With somewhat poorer ores, or with material crushed to 1 in., a stronger current, say, of 7 ampères, would be required to make a similar quantitative division of the material.

Wenstrom Separator.—The Wenstrom, one of the earliest separators to receive extended use and still largely employed in Sweden, is likewise a drum separator (Fig. 359). Inside a cylindrical drum built of alternate wood and iron bars laid along the face, magnets of circular shape are fixed independently of the drum and eccentrically to it. Pole extensions from these magnets, shaped to the drum curvature, carry the magnetic flux to the iron bars of one vertical half of the drum, the other half being relatively removed from the magnetic field. The windings around the magnets are such that alternate extensions have opposite polarity, this disposition of the poles again securing a short magnetic circuit and consequently a high induction. A drum of 30 in. diameter and 24 in. face, making 30 revolutions per minute, and supplied with, say, 15 ampères of current has a capacity of 5—10 tons of coarse ore per hour.

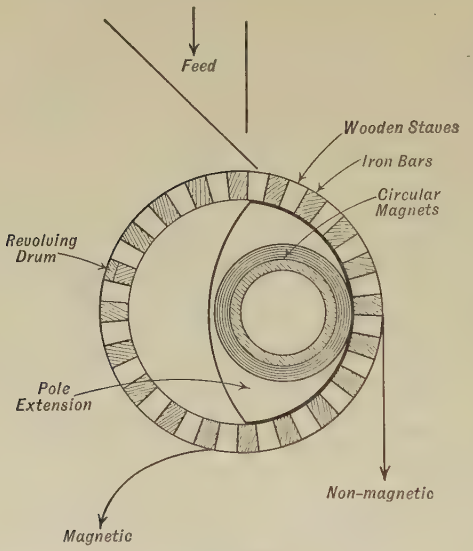


Fig. 359.

Wenstrom Separator.—Diagram. A drum separator used largely in Scandinavia wherever the magnetite is coarse enough (p. 530).

Ball-Norton Pulley Separator.—With this separator the field is obtained by radially-disposed magnets filling the inside of a pulley, around which and around an idler pulley about four feet away, centre to centre, runs a rubber belt on to which the ore is fed. These pulleys are about 30 in. diameter and 24 in. face, and make about 50 r.p.m., the magnets, of which there are usually about twenty-four, revolving with them. As before, the magnets are wound so that poles of opposite polarity follow one another (Fig. 360).

Energized with 20—25 ampères, these separators have for their particular province the treatment of lean ores, or the middling from a previous cobbing operation. Treating 20—30 tons of such material per hour, they produce an enriched middling for further treatment and a tailing assaying, say, 8 per cent of iron. Such further treatment of the middling would generally consist of crushing by rolls to 1 in., followed by treatment on a cobbing drum.

Ball-Norton Belt Separator.—The belt separator is applied to the treatment of material crushed to $\frac{1}{4}$ in. and less; for instance, the middling from the cobbing drum just mentioned. With it, a rectangular battery of a dozen magnets of alternately opposite polarity is contained between the upper and lower stretches of an 18—24 in. belt running at a speed of about 360 ft. per minute around two pulleys, the poles facing downwards close above the lower stretch (Fig. 361). Close below this belt a second and similar belt, set a little back yet between the same side-planes, runs at the slower speed of about 200 ft. per minute in the same direction. This second belt is the feed belt by which the material is brought into the magnetic field. Arrived there, the magnetite, flying upward toward the magnets, is caught by the fast-moving upper or 'take-off' belt, to be dropped when the magnets are passed, at a point well beyond the end of the feed belt, that is, beyond the falling point of the non-magnetics.

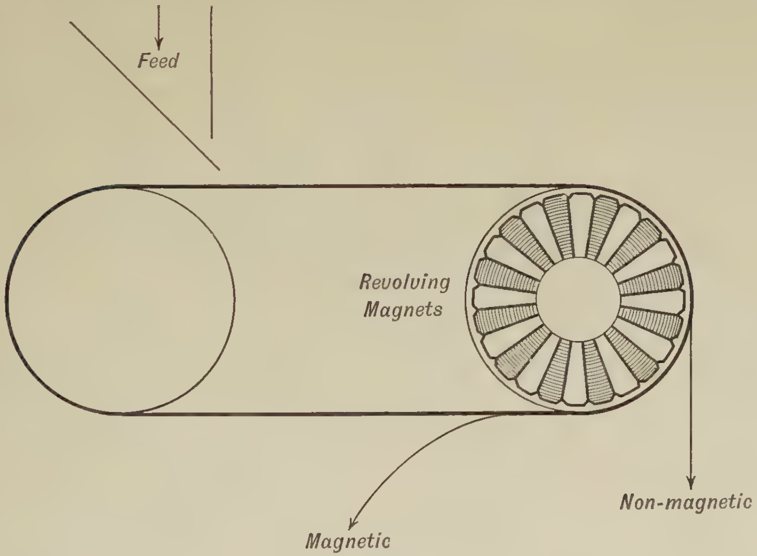


Fig. 360.

Ball-Norton Pulley Separator.—Diagram (p. 530).

Usually these non-magnetics are retreated by another belt machine below, this second machine producing middling to be recrushed and retreated, and a tailing poor enough to be discarded. Two machines so placed in regard to one another would be working in retreatment series (Fig. 383). With still smaller material,, from which the taking of a middling to be recrushed might be unwarranted, the treatment in series would not be applied, but where more than one machine was required the machines would work in parallel, each taking an aliquot portion of the material and making finished products, concentrate and final tailing.

Supplied with about 25 ampères of current and treating $\frac{1}{4}$ in. material, a belt machine will treat about 20 tons per hour, making its particular products, finished or otherwise; the effective capacity of machines working in series is that obtained by numerically dividing the tonnage between the machines in the series. Treating finer material in parallel, say, material smaller than 2 mm., the capacity of a belt machine of normal size is substantially less and about 1·5—2 tons per hour.

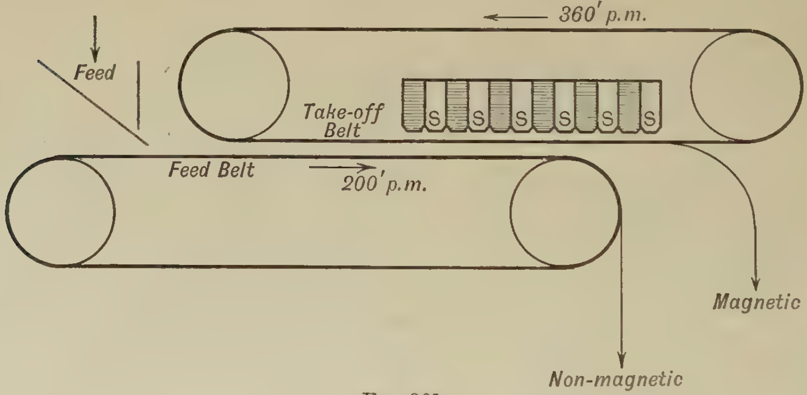


FIG. 361.

Ball-Norton Belt Separator.—Diagram (p. 531).

Gröndal Wet Separator.—The fine crushing of dry material creates considerable dust, which is objectionable not only around the crushers but eventually also around the magnetic separators; moreover, treating such material dry, experience shows that some gangue dust becomes so attached to the magnetite particles as noticeably to diminish the value of the resultant concentrate. Where, therefore, fine grinding is necessary by reason of the intimate mixture of magnetite and gangue, dry separation loses its advantages and wet separation is better.

In Scandinavia many of the magnetites are fine-grained, so that, in addition to coarse and dry separation, wet grinding in ball-mills and subsequent wet magnetic-separation are practised. The ball-mills employed are of the Gröndal and Herberle types, and grinding is taken to 1·5 mm., or, say, 8 mesh, and finer. The resultant pulp, having a consistency of about 4 of water to 1 of ore, is then treated in wet separators, mostly of Gröndal design (Fig. 362).

The Gröndal wet-separator commonly consists of a non-magnetic drum revolving around a fixed magnet with a wedge pole pointed downwards. Upon the exterior of this drum, strips of iron are laid along the face at close intervals around the circle, these strips becoming magnetized as they revolve past the pole. Such revolution takes place at the rate of about 75 revolutions per minute, over a pointed box into which the pulp is fed and maintained at such a level that the drum just breaks the water surface. Arrived in this box the solid particles in the pulp are lifted upwards over a central baffle by water entering below. Above the baffle they come into the magnetic field, the magnetite particles attaching themselves to the magnetized strips, the non-magnetics falling to a bottom discharge. Rising out of the water the magnetics are carried out of the field, till, the strips losing their magnetism, the particles drop, the middling first and the cleaner particles afterwards, water being used to complete the detachment of the latter. By trays suitably placed, these two products may be collected separately, but generally they are collected together as a middling to be retreated by a second machine in series. Such machines are used at Grangesberg, Dannemora, and elsewhere in Sweden.

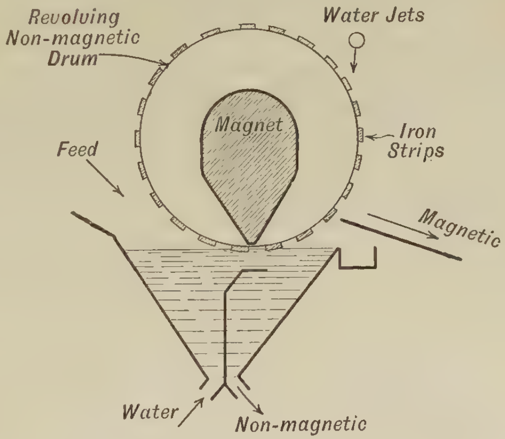


FIG. 362.

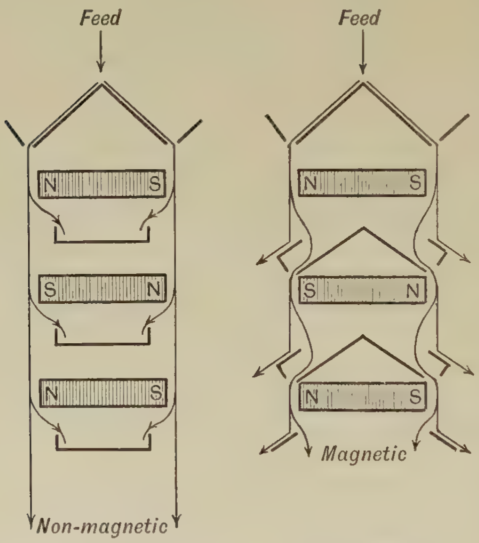
Gröndal Wet Separator.—Diagram (p. 533).

In America the Gröndal wet-separator is modified in that the pointed pole gives place to a battery of magnets arranged within the drum on its rising side, the drum itself having no iron strips on its face.

The property of magnetite slime in suspension to become selectively flocculated in a magnetic field permits its separation from gangue slime by descent through a rising current strong enough to carry the gangue slime to the overflow. Such a separation of the gangue slime may be carried out beforehand to the greater cleanliness of the concentrate, or in the magnetic separator itself.

Magnetic Log-washer and Magnetic Drag-classifier.—By placing magnets close under the sloping bottom of a log-washer, a wet magnetic-separator suitable to the treatment of finely-ground magnetite ore, and of large capacity, is obtained. Recently, in the Eastern Mesabi, Minnesota, such a magnetic log-washer was used with technical success in treating magnetite ore previously ground in wet ball-mills to 100 mesh and finer. The fine magnetite, bunched together over the magnets, becomes involved in the upward-conveying effect exercised by the spiral-flighted logs, while the gangue is washed downwards eventually to overflow.

It has also been suggested that the drag-classifier may be converted to a similar use.



Roughing Machine **Cleaning Machine**

Fig. 363.

Edison Deflection Separator.—Diagrams of roughing and cleaning machines (p. 534).

Edison Deflection Separator.—Edison designed and put into use upon finely-crushed dry magnetite ore, a separator consisting solely of a battery of horizontal magnets, four or five in number, placed at regular intervals one above the other (Fig. 363). So placed, each end of each magnet becomes a separating field, in which the magnetic particles of a falling sheet of ore are deflected towards the magnet sufficiently to be withdrawn from the falling stream. This happening in succession from top to bottom, the material which maintains a straight fall will be fairly clean gangue. That this shall be so, however, the catch plates must be so placed that lean material is caught with the concentrate. To clean this rough concentrate, a second battery of magnets is arranged, with the catch plates suitably disposed to separate a portion of the lean mineral each time, the enriched mineral alone being directed past the poles in succession.

Such a separator includes no moving parts and is remarkably simple. It demands, however, that the material shall be dry, of uniform size, say crushed to 8—25 mesh, and contain a minimum of dust. With poles flattened to accommodate a stream about four feet wide, and using 10—15 ampères, a single separator will treat about 10 tons per hour. In the roughing machine the windings are arranged that the fields increase in strength downwards, whereas in the cleaning machine the strongest field is at the top. The magnets, being enclosed, are not naked to the stream.

This separator was tried both at Edison, New Jersey, and at Dunderland, Norway.

The separators for magnetite just described were distinguished by simplicity, robustness, and capacity, rather than by perfect work. More highly developed are those applied to the treatment of a roasted pyritic-concentrate, generally to remove pyritic impurities from blende; such separators are:

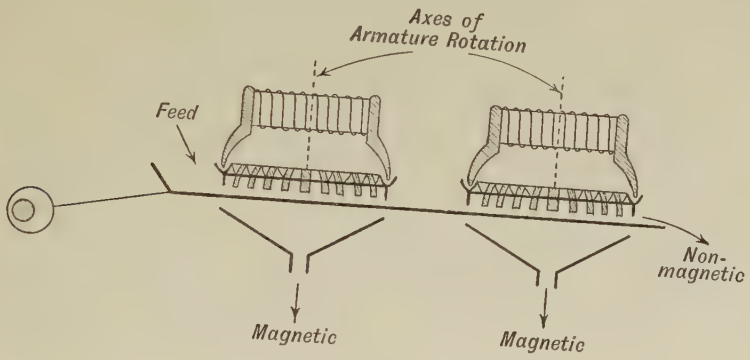


Fig. 364.

Dings Rougher Separator.—Diagram. The studded revolving-armatures have a diameter of about 26 inches, so that they overhang and throw the magnetics to either side of the table, this latter being about 21 inches wide (p. 535).

Dings Separator.—In the Wisconsin zinc-district of the Upper Mississippi valley, two types of Dings separator are used, a rougher and a finishing machine. The rougher machine consists of a vibrating table about 21 in. wide and 8 ft. long, shaken endwise by two eccentrics making about 450 revolutions per minute, and set at an inclination of about 4 in. per foot (Fig. 364). This table has a pressed-asbestos top to accommodate the still-warm material which it treats. Above, and aligned with it, are two pairs of magnets, the first pair having a shallower winding than the second, all being about 20 in. in length. The poles of each pair, extending downwards and outwards to end in wedges, follow the circle of an armature ring revolving in a plane parallel to the table. At regular intervals of about $1\frac{1}{2}$ in. around this ring, which otherwise is of brass, are small vertical iron-studs which extend to within $\frac{3}{8}$ in.—$\frac{1}{2}$ in. of the table surface. These studs when under the poles become the effective though secondary magnets of the separator; to conserve the magnetic flux of the primary poles, they branch upwards to form a fork by which the primary pole is embraced. Revolving from under the poles, the diameter of the ring is such that at the neutral axis of the magnets these studs drop the magnetics clear of the table, into appropriate chutes. Employing 2 ampères of current in the first pair and 3 ampères in the second, this separator is capable of treating 2—2·5 tons of concentrate per hour, removing magnetic sulphide principally, but also some magnetic oxide. In the place of the vibrating table a conveyor belt is sometimes used, the other items of design remaining the same.

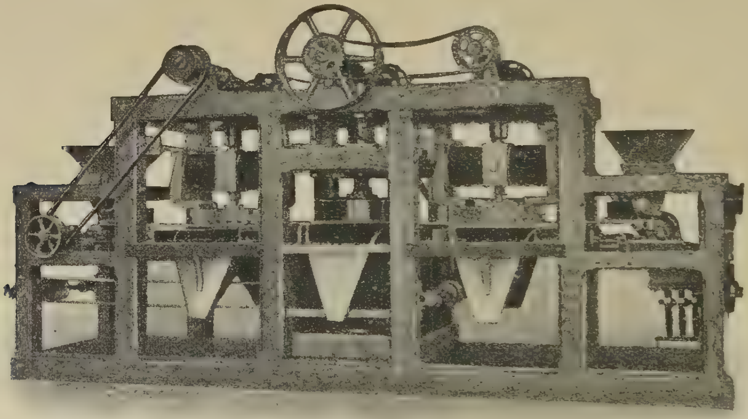


Fig. 365.

Dings High-Intensity Separator.—General view. The illustration shows two short feed-belts moving inwards towards the centre of the machine, one from either end. Over each belt is a large magnet, the poles of which extend downward to energize an iron-studded revolving armature by which the magnetics are lifted from the belt and dropped at the side. In addition, a solid armature revolves over the discharge-ends of the two belts to remove a magnetic middling from each; this armature is energized by magnets below the belt. The two non-magnetic discharges are caught below on a small belt which carries them clear of the machine (p. 536).

The Dings finishing separator is a belt machine with a field strong enough to remove a magnetic middling from the non-magnetic product of the rougher machine. The magnets now are placed below the belt, the revolving armature remaining above, with an air-gap of about $\frac{3}{4}$ in. between (Fig. 365). The armature being a solid steel disc, except for the air-gap the magnetic circuit is completely through iron, and the induction is high;

moreover, the magnets have deep windings and are supplied with 7—15 ampères of current. Doing its particular work a common capacity for this type of separator is about one ton per hour.

A Dings wet-separator has also been developed. Such a separator was used at Trail, British Columbia, to remove pyrrhotite from galena and blende, the ore treated being that from the Sullivan mine. Dry magnetic separation having failed because of adhering dust, the ore was ground wet in ball-mills to 100 mesh, and then treated in the condition of a 4 : 1 pulp, on a flanged endless belt 4 ft. in width and about 6 ft. in length. Over this belt and aligned with it was a wound magnet, the two poles of which spanned the belt, one in an upstream and the other in a downstream position. Running between each pole and the table was a transverse take-off belt. Below the main belt an iron bridge-bar completed the magnetic circuit. In addition to permitting the use of wet ball-mills, water was found to be of benefit in the actual separation, in that it secured an even stream across the belt, and washed the magnetic particles free of adhering gangue as they flew out of the water to the take-off belt. At the Sullivan mine, five of these separators, 4 ft. in width, shared the primary stream of ore coming from the ball-mills, each machine treating about 100—120 tons per day. In retreatment thirteen similar machines were employed for a total capacity of 600 tons per day.

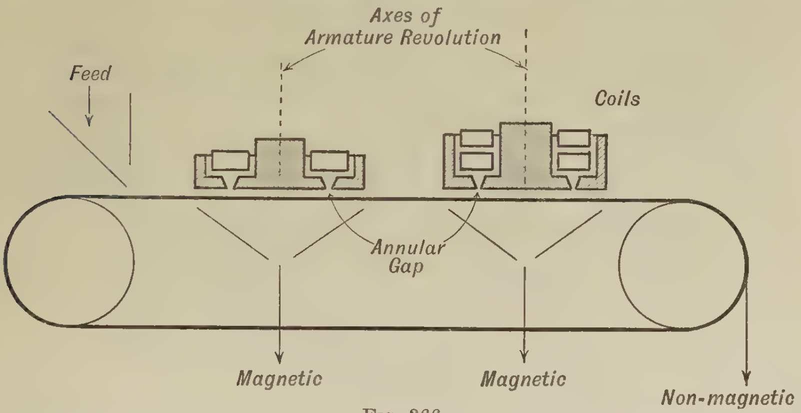


FIG. 366.

Cleveland-Knowles Separator.—Diagram. The core of the first magnet is of cast-iron and the windings are light ; the core of the second magnet is of steel and the windings are heavy (p. 537).

Cleveland-Knowles Separator.—This separator, again a belt machine, is likewise much used in the Wisconsin district. Over an endless belt moving about 100 ft. per minute, two disc-shaped magnets, of greater diameter than the belt width, revolve at a speed of about 40 revolutions per minute

(Fig. 366). These magnets consist of an inner wound-core and an outer shell, shell and core being separated by the windings and a wooden filling. On the bottom, however, shell and core meet except for an annular gap of half an inch, filled with metallic zinc. Across this gap the magnetic circuit completes itself, and here the magnetic field obtains. From the belt about 1 in. below, the properly-roasted marcasite and pyrite rise to the magnet and particularly to the annular gap, across which they bridge themselves, this attachment being broken as the magnet revolves against a scraper beyond the belt. The first magnet is wound lighter and given less current, say 2 ampères, its duty being to remove the strongly magnetic particles; the second magnet is wound heavier and given say 4—10 ampères, to remove a less magnetic product. Treating ordinary blende-concentrate, this separator has a capacity of about 1 ton per hour with a 12 in. belt and 2 tons with a 21 in. belt.

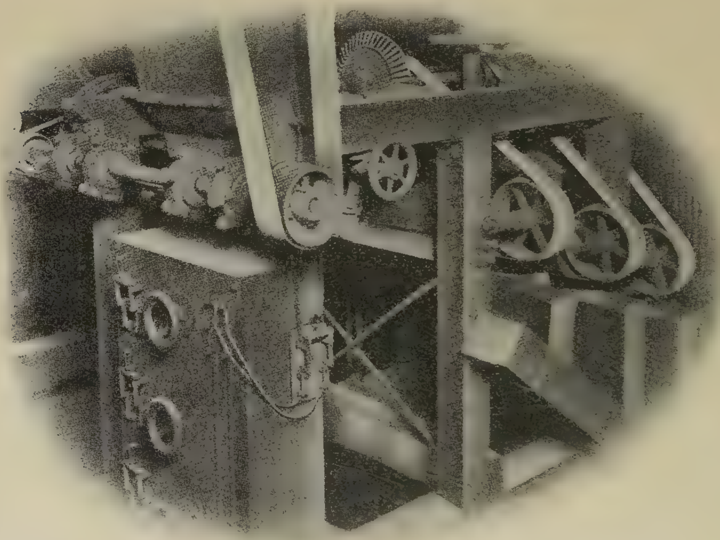


Fig. 367.

Campbell Separator.—General View. The illustration shows an inclined table shaken by two eccentrics behind. The three cross belts serve three magnets, respectively. The electric-control fittings of these three magnets are seen in the foreground (p. 538).

Campbell Separator.—This separator, like the Dings roughing machine, has a shaking inclined-bed; the magnetic products are, however, removed by cross belts (Fig. 367).

Where an enriched product from water concentration contains as preponderating impurity a mineral which by roasting may be made strongly magnetic, a wet separator, without sizing, may satisfactorily treat material containing all sizes below say 8 mesh. Such conditions obtain, for instance, with the pyritic middling from jigs and tables at the Llallagua tin mine, Bolivia, this middling consisting of about two-thirds of pyrite, the remaining third being largely of cassiterite; no wolframite is present, and scheelite but in unimportant amount.

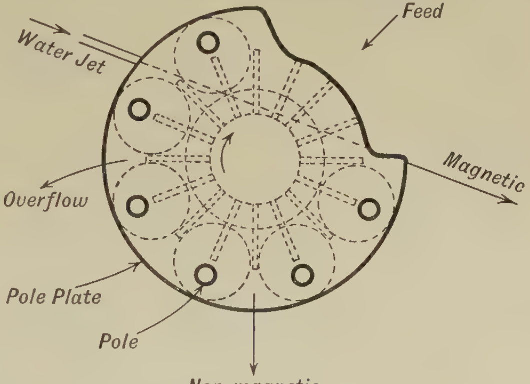


FIG. 368.

Stern Wet Separator.—Diagram. Six cylindrical magnets arranged parallel to one another are disposed three parts round a circle, the other quadrant remaining unoccupied. The magnets are fixed, their poles ending in the two end-plates of the machine. Within the internal space thus provided revolves an iron core having starred ends; between these starred ends and the end-plates are the two fields or separating chambers. Fed midway between the end-plates, the pulp flows into the separating chambers, where the magnetics attach themselves to the star points, bridging the gap between these points and the end-plates, while the non-magnetics fall to a bottom discharge. Carried upwards and out of the water by the rotation of the iron core, the magnetics are detached by a jet of water (p. 539).

Stern Separator.—This separator has six magnets disposed as though around three parts of the face of a cylinder, the remaining quadrant being clear and open. In that position they are held by cast-iron end-plates, between which and themselves a trough about 3 feet long is contained. These end-plates become the pole plates of the machine. Revolving between them and concentric with the cylindrically disposed magnets is an iron core which at its ends throws out radial arms to form star-shaped flanges. Between these star flanges, which have an outside diameter of about 24 in., and the pole plates, are the two magnetic fields, one at each end. Admitted through the open quadrant into these fields, the roasted ore, suitably borne in about three times its weight of water, comes under treatment, the magnetics flying to the points of the star while the non-magnetic cassiterite drops to a spigot discharge. Attached to the star points, the magnetics in the course of revolution arrive above water-level where, in the open quadrant, they meet a water jet which projects them into the discharge chute.

At a speed of about 15 revolutions per minute and using about 6 ampères of current, this separator treats about 1 ton of ore per hour.[1]

Finally, before leaving the description of ferro-magnetic separation, it deserves mention that, until recent years, siderite, commonly occurring as a gangue-mineral in blende ores on the Continent, was removed from the water concentrate in which it accompanied the blende, by a magnetic treatment after roasting, the separators employed being generally of the drum type. To-day, this separation is made in fields of such strength that no roasting is necessary; and the separators providing these fields come within the description of feebly-magnetic separators.

Feebly-magnetic Separators.

The earliest machines of this class were those designed by Wetherill to treat the manganiferous zinc ores of Franklin Furnace, New Jersey. These machines, which came into successful operation about the year 1895, embodied the discovery of Wetherill, that, in the strongly attractive field around a pointed pole, a number of minerals previously not considered by the miner to be magnetic, could readily and surely be withdrawn from a passing stream of dry fine material.

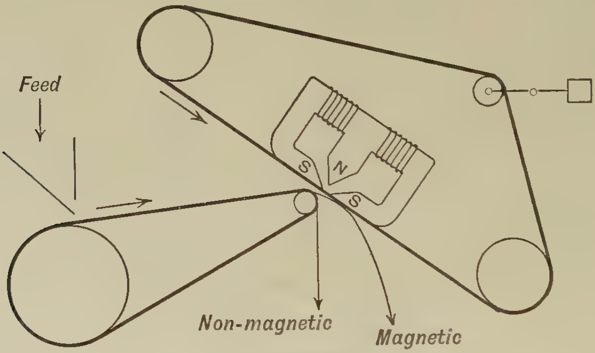


Fig. 369.

Wetherill Separator (Early Type).—Diagram (pp. 524, 540).

Wetherill Separator.—In early types of these separators, the field was formed by two pointed south-poles bent round almost to meet a pointed north-pole centrally disposed between them and common to them both (Fig. 369). Around this magnetic system ran an endless belt which bore upon the outward faces of the south poles and turned around three pulleys triangularly disposed, one being a tension pulley. The system was also so reclined that the stretch of belt bearing upon the poles moved downwards at an inclination through the field. So canted, the feed was carried into the field by another endless belt running-in beneath, which belt, turning over to return empty, left the material for magnetism and gravity to dispute control. The magnetic material, defying gravity and flying towards the poles, met and travelled with the intervening belt, to be dropped when out of the field; the non-magnetic material on the other hand dropped early. In this design the material treated did not pass between, but to one side of the poles.

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[1] Copeland and Hollister, *E. & M.J.*, September 25, 1915, p. 513.

The present designs of Wetherill separator all include passage between opposing poles, the lower of which is flat and gives support to the loaded belt, the upper pole being pointed (Figs. 370, 371). To the development of this design Rowand contributed. The feed moves forward on a flat belt 10 in.—18 in. wide running around two end pulleys, by one of which it is driven at a speed of 30—100 ft. per minute. Down the length of this feed belt, two or more rarely three pairs of magnets follow one another, each pair consisting of a magnet below the material facing another above, the magnet below having two flat poles turned upwards, that above having two pointed poles turned downwards. As the two magnets of a pair are wound so that poles of opposite polarity face one another, and as also the air-gap between these poles can be brought as low as $\frac{1}{2}$ in.—$\frac{3}{4}$ in., the magnetic circuit is well favoured and induction is high. With two pairs of magnets, the second pair are wound to obtain a higher field, that is to say the number of wire turns per centimetre is greater; in addition, while the iron core of the first magnet is 15 in.—18 in. long, that of the second is usually 18 in.—24 in., this difference intensifying the field. All the poles extend across the stream, their length in this direction being about an inch less than the belt width.

Between each upper pole and the material on the feed belt is a narrow belt running transversely to the stream; against this belt the magnetics strike as they fly upward to the pointed pole, and by it they are carried out of the field to be deposited at the side. These belts are described as cross or 'take-off' belts; they are usually 4 in.—6 in. in width and run at a speed of 100 ft.—300 ft. per minute, each around three pulleys triangularly disposed, two of which are in the horizontal plane of the pole. Where there are two pairs of magnets, it is usual to have the same air-gap at either field of each pair, so that the two cross belts serving such a pair remove a similar product. By lowering the downstream end of the upper magnet it is possible, however, to obtain a stronger field and a different product at that end. This is also true of the second pair of magnets, the upper magnet being again so hung that the air-gap at either end may be adjusted. With but a single pair of magnets, that is, a two-field machine, a smaller air-gap at the downstream end would be necessary if two different magnetic products were desired; on the other hand, where there were three pairs of magnets, that is, a six-field machine, any necessity to vary the two air-gaps of any pair would be eliminated.

Concerning capacity, the rate of feed is largely determined by the size of the material treated, its permeability, and the proportion of magnetics contained. With fine material, say below 40 mesh, a fast speed of the feed belt would necessitate fast speed of the cross belts, and loss and irregular working by dust would result; the depth of material upon the belt must also be less with fine material. Accordingly, and for the further reason that fine gangue tends to stick to the belt, there is difficulty in treating material finer than 150 mesh. The most suitable size appears to be 20 mesh; coarser material appears to demand a slower speed. With greater permeability, the belt speed may be greater, since there is greater attractive force to set against the greater momentum of the faster-moving material. With a larger proportion of magnetics, the rate of feed would be limited by the capacity of the cross belts. With the cross belts running at the maximum speed possible with the particular size of the material,

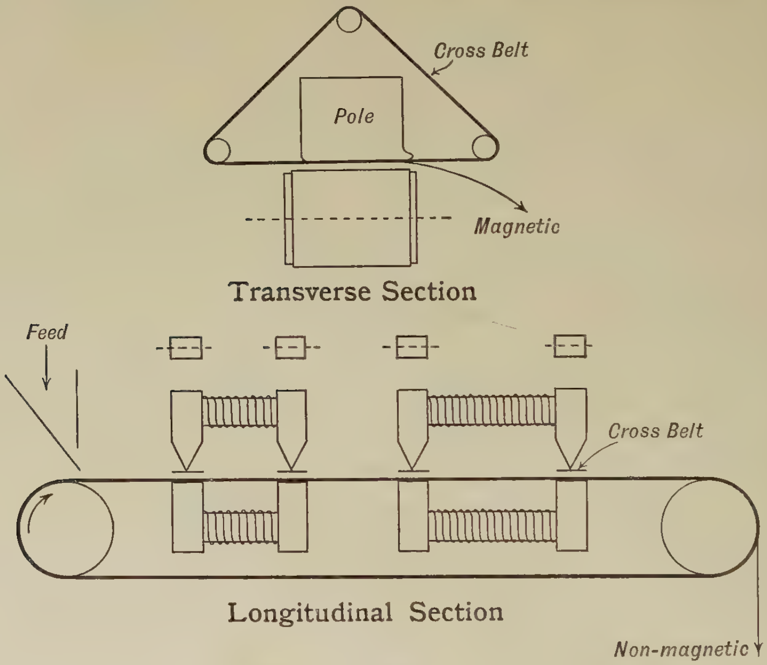
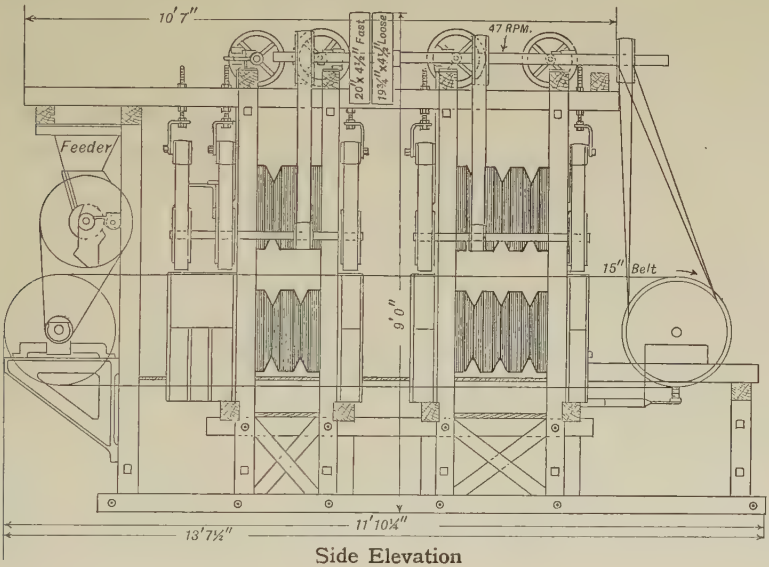


FIG. 370.

Wetherill Separator.—Diagram. Transverse and longitudinal sections of a four-field machine, this being the common type (p. 541).

a relatively slow speed of the feed belt or a narrow feed belt would be proper where the proportion of magnetics was high, say 40 per cent; on the other hand, a wide belt or higher speed would be justified with a lower percentage of magnetics, say 5—15 per cent. All Wetherill separators are provided with step pulleys so that the speed of the feed belt may be adjusted to the capacity of the cross belts. Common figures of capacity appear to vary around one ton per hour with an 18 in. belt and material above say 30 mesh, this capacity falling to about one quarter when treating 60-mesh material.



Side Elevation

FIG. 371.

Wetherill Separator, Side Elevation.—The illustration gives an idea of the depth of the coils and the form they assume for their better ventilation. It is seen that the feeder contains a revolving sector (pp. 541, 550).

Wetherill separators have a wide use. At Franklin Furnace, New Jersey, to remove franklinite from willemite and zincite; in Cornwall, Tasmania, New South Wales, Bolivia, and elsewhere, to remove wolframite from cassiterite; in India, Ceylon, and Brazil, to remove monazite from beach sands containing zincon, quartz, etc.; on the Continent, to remove siderite from blende.

Rapid Separator.—Instead of employing a cross belt to take the magnetics out of the field, the magnet above the feed belt may be used for this purpose, provided it be in the form of an iron disc rotating about a vertical axis over fixed magnets beneath; such, in general, is the design of the Rapid magnetic separator (Figs. 372, 373). The disc of this machine is not massive like that of the Cleveland-Knowles, nor studded like that of the Dings, but in the shape of an inverted saucer, the rim of which, extending downward, picks up the magnetic material to carry it off and drop it out of the field, at the side. Such a disc has an advantage in that it is stiff and cannot develop slackness; accordingly, and also because there is no intervening cross-belt, the air-gap may be smaller; on the other hand, being a rotating member it is not conveniently wound, and therefore not separately energized. The disc being wider than the feed belt, its rim during rotation crosses that belt twice, the second crossing being made in an opposite direction to the first; at each crossing it is operative, discharging a product first to one side and then to the other. If, instead of being set normally, the disc be slightly canted endwise, a departure which its support allows, more-strongly magnetic material may be separated at the first crossing, and less-strongly magnetic material at the second. The separation of minerals of different magnetic permeability is, however, more surely obtained by placing two or three discs in series over magnets successively stronger. These magnets are usually so disposed that a pole comes between two discs, serving them both, while an additional

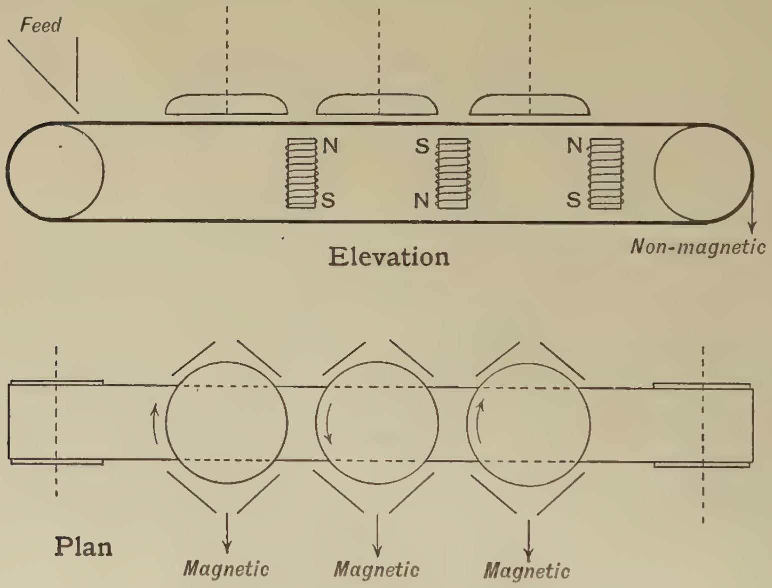


FIG. 372.

Rapid Separator.—Diagram. Of a 3-disc machine suitable, say, for monazite sands, etc. (p. 544).

pole serves the downstream side of the last disc; more rarely another serves the upstream side of the first disc. All these poles are flat, the lines of force converging from them to the disc-rims, which thereby become strong secondary poles.

Separating wolframite from cassiterite with this machine, coarse wolframite is removed by the first disc, the fine remaining till the last. Treating Brazilian monazite sands, a spectacular separation is made of black ilmenite, red garnet, yellow monazite, and white quartz.

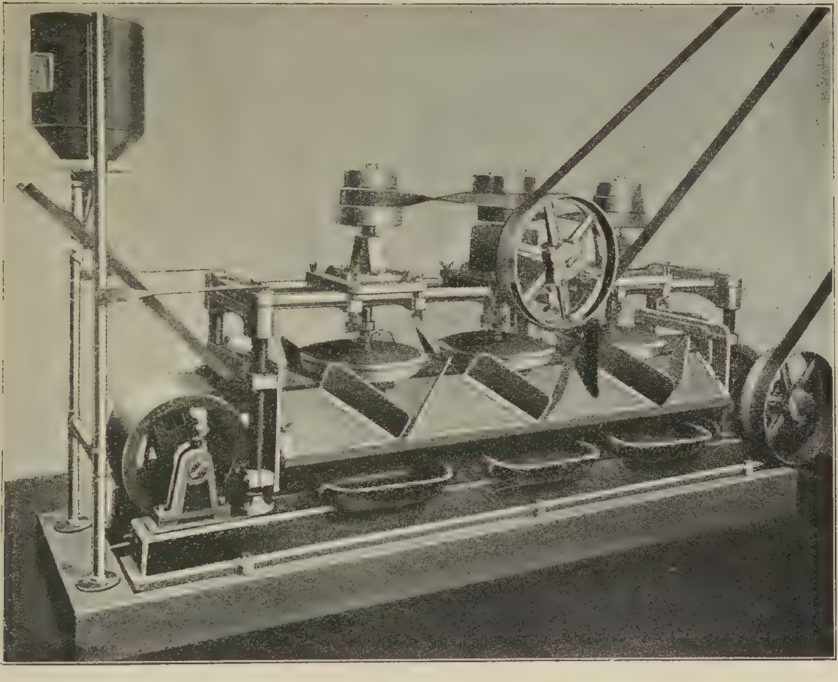


Fig. 373.

Rapid Separator. — General View of a 3-disc machine. The feeder of this machine consists of a metal bin, with a glass funnel below. The bin has a conical bottom with a relatively large discharge-aperture at the apex. The smaller aperture of the funnel not being able to pass all the material which would freely run out of the bin, the material builds up in the funnel till the discharge from the bin is restrained. By changing the funnel for another with different aperture, the rate of feed is altered (p. 544).

The feed belts of these machines are 6 in.—15 in. wide, the current 2—12 ampères, and the capacity 0·05—0·25 ton per hour.

Ullrich Separator.—Employing the same idea of flat poles beneath and pointed poles above, the Ullrich separator differs from the two first described in that all the pointed poles are over one flat pole, the different field strengths being obtained by arranging the succeeding pointed-poles successively lower (Fig. 374). With this compact assembly of the poles the path of the material through this machine is short, so short that many separating zones, six or eight for instance, may be arranged radially around a circle of, say, 8 ft. diameter (Fig. 375). In such a design only the flat poles are energized, the two at the ends of a common diameter to an opposite polarity; these magnets and their windings are fixed. The pointed poles, on the other hand, are rings of soft iron rotating around the circle, their support completing the magnetic circuit across the circle; above each flat pole, the pointed poles, becoming energized, pick up their load, which they carry off but quickly drop, a wiper at the neutral zone half-way to the next separating zone, ensuring complete detachment.

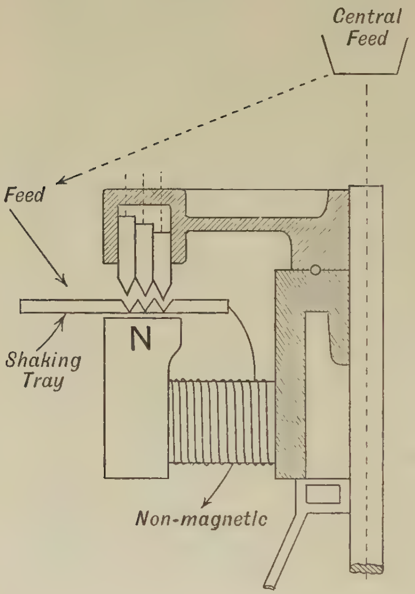


FIG. 374.

Ullrich Separator. — Diagram. The magnetics are caught as they stream off the rotating rings. The air-gap of each pointed ring may be adjusted separately (p. 545).

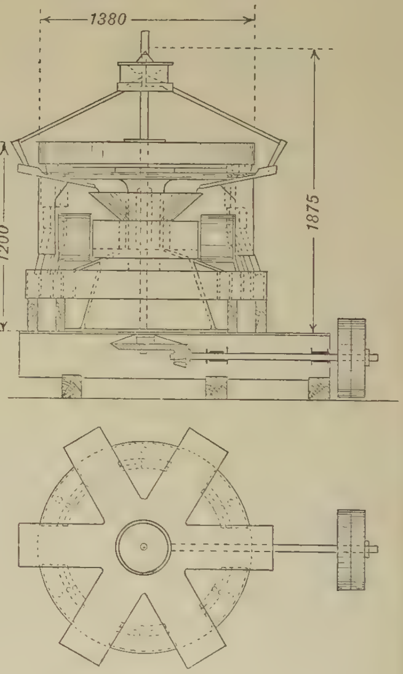


FIG. 375.

Ullrich Separator.—Plan and elevation of a six-zone machine. These views show the arrangement of six zones around the circle, and the central feed suitable for wet and dry material. The figures given are dimensions in millimetres (p. 545).

Meanwhile, the non-magnetic material moves radially inwards to fall into a central chute, its progression being effected either by a short endless belt or by a shaking tray. The ordinary width of the stream is 6—8 inches, and the length perhaps a foot.

The Ullrich separator is largely used on the Continent to separate siderite from blende; treating 5-mesh material it has a capacity of about 0·2 ton per separating-zone per hour. It will also satisfactorily treat monazite sands at the rate of about 0·05 ton per zone per hour.

By arranging a flow of water down the rings, on to the sloping tray along which the pulp is streaming, this separator has also been applied to the treatment of an ore-pulp, fine magnetite, for instance; the water being continuous across the air-gap it is considered that the material in flying to the magnet is under no necessity to break the water surface, and that therefore its flight is not hindered by the surface tension of the water. This design has not, however, led to the wider adoption of wet magnetic separation (Fig. 376).

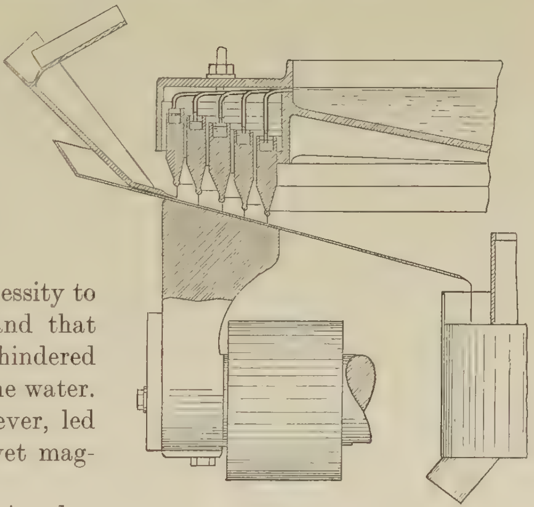


FIG. 376.

Ullrich Wet Separator.—Diagram. The use of this separator is only possible with ferro-magnetic material; the air-gap is constant though five rings are shown (p. 547).

Mechernich Separator.—Another design giving a strong and concentrated field is that which includes two magnetized cylinders placed parallel and close to one another, the lines of force being thereby concentrated in the plane containing the axes of the two cylinders. This design is embodied in the Mechernich separator, which has two magnetized drums laid parallel, and with the plane containing their axes inclined a little from the vertical (Fig. 377). Of these two magnets, the magnetic circuit of which is completed through their end supports, the upper revolves. Against the lower surface of this magnet and in contact with it, the material to be treated arrives from the over-lying side. From that position the non-magnetics drop to the lower cylinder, from off which they fall on the over-hanging side. The magnetics, on the other hand, attaching themselves to the upper cylinder, are held against that cylinder till carried out of the field, when they fall, to be caught on suitably disposed aprons. This separator has been largely used on the Continent; it was also tried at Broken Hill, N.S.W., to remove rhodonite from blende. It possesses the advantage of permitting direct contact of the material with the naked pole, but has the disadvantage of a revolving magnet. On mines where other separators are also in use, the fine material has sometimes been apportioned to this separator.

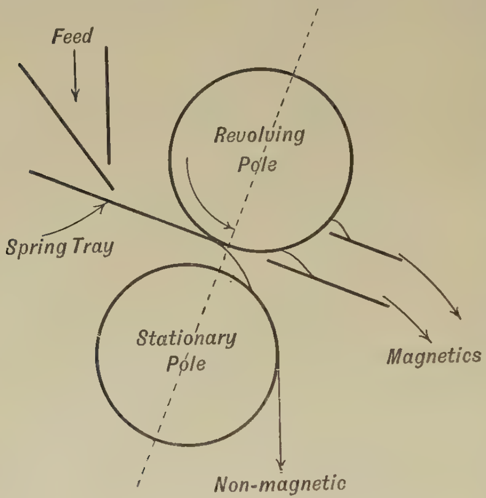


FIG. 377.

Mechernich Separator.—Diagram (p. 547).

Motor Separator.—A somewhat similar separator, but one with stationary magnets, is that known as the Motor separator, because between two magnetic poles a cylindrical armature suitably wound is rotated by the passage of current (Figs. 378, 379). This armature, being inductively magnetized by the stationary magnets, takes the place of the revolving magnet of the Mechernich separator, separation proceeding similarly; to ensure attachment to the armature in preference to the fixed pole beneath, the armature is built of alternate discs of iron and non-magnetic material, the former projecting. This machine, requiring no mechanical drive, is convenient. It is used on the Continent to separate siderite from blende, in Nigeria to remove iron-minerals from alluvial tin ore, and elsewhere. At Broken Hill it separated rhodonite in a more magnetic product, blende in a less magnetic, and galena and quartz in the non-magnetic product.

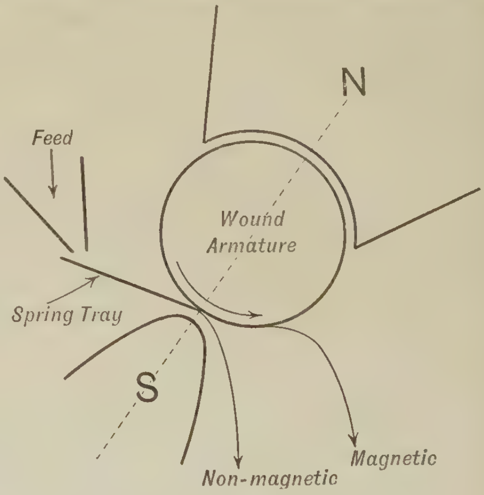


FIG. 378.

Motor Separator.—Diagram (p. 548).

International Separator.—Lines of force will also concentrate upon a cylindrical armature placed between two poles if the surface of that armature be studded with points. An array of such points is the feature of the International separator the armature of which is mechanically revolved around a horizontal axis between two oppositely-magnetized poles (Fig. 380). The pole on one side is recessed to envelop the armature above and below its axis, while that on the other side is cut away below the axis. Fed from on top, the material is carried by rotation to this other side, whence it passes in a thin stream between armature and pole. Reaching the level of the axis the non-magnetics fall vertically; the magnetics, sticking to the points, fall or are thrown-off later.

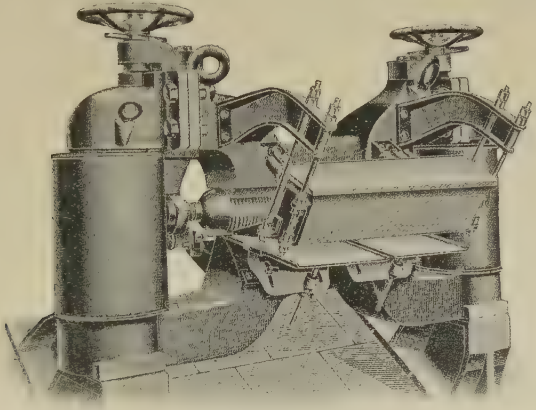


Fig. 379.

Motor Separator.—General View. This view, which is from the feed side, shows very well the magnetic poles with the armature between. The striped appearance of the latter indicates the alternation of discs of iron with discs of non-magnetic material, the iron discs forming ridges (p. 548).

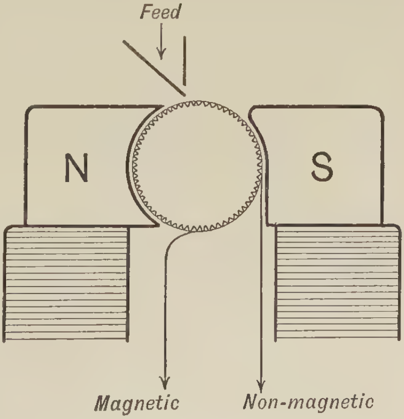


Fig. 380.

International Separator.—Diagram (p. 549).

This particular separator was tried at Leadville, Colorado, and at Broken Hill, to separate magnetic blende; and elsewhere.

The current used in magnetic separators is direct current of a pressure variously from 70 to 250 volts; that in use for lighting can generally be conveniently used for magnetic separation. Low voltage may be used when the windings are of thick wire. When the windings are enclosed, as they often are, a large wire is necessary to prevent undue heating. A large wire is also necessary when the separator has to work continuously, since continuous running causes the temperature to mount (p. 522, Fig. 371).

Neither in voltage nor in amount of current does there appear to be any difference in the respective provisions for ferro-magnetic and feebly-magnetic separators; the stronger fields of the latter are obtained by making the magnetic circuit completely of iron—except for the air-gaps (Fig. 353); by pointing one pole or providing it with points; and by longer or by deeper windings.

Regulation of the field strength of individual separators may be, however, made by adjusting the amount of current; for this purpose all separators, and sometimes each magnetic circuit of a multi-circuited separator, are provided with ammeter and rheostat (Fig. 367). In addition, the mechanism of the separator may also be adjusted to a particular separation; thus, the air-gap may be opened wider or made more narrow; the feed belt or drum may be run faster or slower, as also may the take-off devices where such exist. Finally, the depth of the stream of ore brought into the field may be suitably varied.

Alternating-current Magnetic Separation

Investigation of the possible use of alternating current in magnetic separation, either in the direction of obtaining a rotary field by polyphase currents, or otherwise, has hitherto not resulted in any useful discovery. Recently, however, W. M. Mordey, by arranging poles energized by two-phase currents to follow one another across the stream, has succeeded in driving iron minerals and iron compounds in that direction (Fig. 381).[1] This effect is not one of simple magnetic attraction and repulsion, but apparently a display of a 'hysteretic repulsion,' consequent upon the magnetism residual after each alternation, this repulsion being made continuous by the moving field contributed by polyphase current.

A laminated alternating-current magnet behaves towards magnetite or iron-filings much like a direct-current magnet, in that tufts of these materials collect at the poles, from which poles lines of force radiate. On the other hand, towards such a feebly-magnetic mineral as haematite no attraction appears to be exercised, but a decided repulsion, manifested, for instance, when a dish containing powdered haematite is laid upon an upturned pole; and made continuous when the dish spans a number of poles energized by polyphase current. Similar repulsion of magnetite occurs at a lower excitation, or when the dish is lifted sensibly off the poles.

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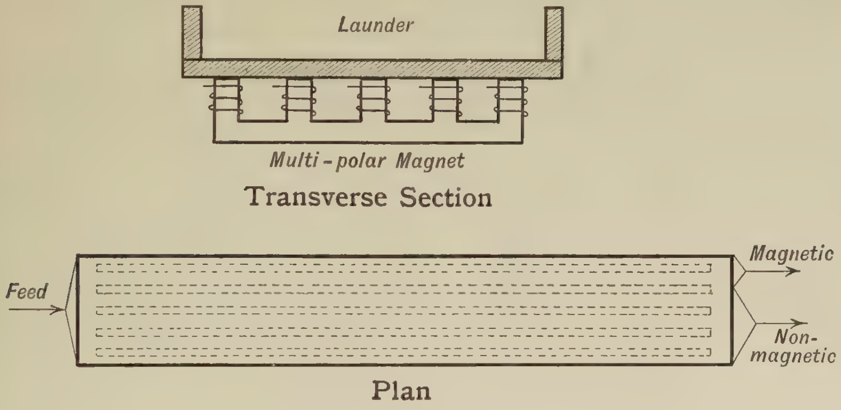
[1] Mordey, *Trans. South African Institute of Electrical Engineers*, December 1921.

From these phenomena it seems probable that with alternating current ordinary magnetic attraction and hysteretic repulsion together determine the behaviour of particles present in the field; of these two factors the former has already been discussed, it remains to indicate one or two features of the latter. Hysteretic repulsion is low and attraction relatively high when the frequency of alternation is low, and *vice versa*; Mordey found that with increase of frequency from 25 to 75 periods the speed at which the material was repelled increased approximately as the square of the increase in frequency. At higher frequencies, however, repulsion again appears to be inactive; Mordey, for instance, found that both at 150 periods and at 350 periods attraction was pronounced, even haematite remaining over the poles. He used relatively low inductions, 560 to 2000, these being more proper to alternating current than the higher inductions associated with direct current in ordinary magnetic separation.



Transverse Section

Plan

Fig. 381.

Alternating-Current Separation.—Diagram. A design which carries Mordey's method of magnetic separation into effect, is illustrated. The magnetics can be made to move up a slight incline transversely to the stream. A fairly long trough or launder would be required (pp. 550, 552).

The continuous repulsion of the ferriferous particles across the stream is forceful and unhesitating, whether these be dry or borne in water; it is assisted by an upward repulsion which frees those particles from entanglement with associated gangue, and gives them power to climb an inclination, or even the sides of the containing vessel. At the same time, however, these particles, and particularly those of magnetite, tend to be strongly held in the plane of their movement, so that, unless the field be properly adjusted, transverse walls or banks will form, and the removal of the gangue downstream will be hindered.

To make use of this discovery Mordey has in mind a shallow inclined launder down which the material would flow in the condition of an ore pulp (Fig. 381). With poles running the length of this launder the ferriferous particles would be driven to one side, so that they could be separately collected at the bottom, the gangue keeping a straight path.

It is interesting that, while simple iron-minerals have shown themselves capable of making the transverse movement, such moderately-magnetic complex iron-minerals as ilmenite and wolframite do not move ; also that, though magnetite moves more strongly, haematite can hardly be said to be outclassed ; and that a small contamination with iron oxide causes other minerals to move, wolframite and cassiterite, for instance. Obviously, therefore, though magnetic susceptibility is doubtless involved, it does not enter unfettered ; as already stated, probably it is associated with hysteretic repulsion. That the repulsion may be due to eddy currents set up in the particles appears to be excluded, from the fact that the conductivity of haematite is not high enough to permit any pronounced development of such currents ; moreover, particles of metallic aluminium, the conductivity of which is very high, are not repelled.

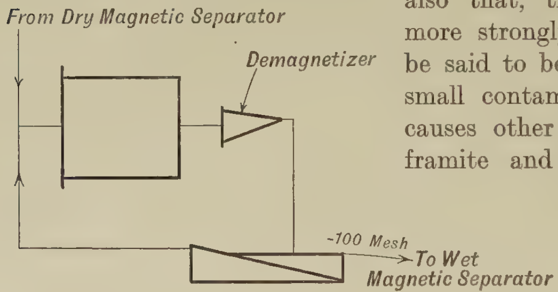


Fig. 382.

Alternating-Current Demagnetizer. — Diagram. The position of such a demagnetizer, in the closed circuit between ball-mill and drag classifier, is indicated (p. 552).

It is the hope that this process of magnetic separation may so develop, that deposits, such as that at Dunderland, Norway, which contain much haematite in addition to magnetite, and others consisting largely of granular haematite, may be successfully beneficiated. In view of the many deposits coming within these descriptions, and of the fact that the present means of magnetic separation, good as they are for dry work, entirely fail to separate feebly-magnetic minerals from a water-borne pulp, such a simple process would have a wide field.

Demagnetization.—When regrinding in a wet ball-mill material which has already undergone a magnetic treatment, difficulty is sometimes experienced in the circuit-classifier by reason of the magnetic particles bunching together and sinking, to be returned to the mill. This difficulty is removed if the material before it enters the classifier is demagnetized. Such demagnetization may be achieved by passing the pulp through a pipe having around it a wire coil served with alternating current. Such a coil is deep at its upstream end and shallow at the other, so that as fixed around the pipe it is of conical outline (Fig. 382). Experience shows that with 50-mesh material demagnetization is hardly necessary, but that with material finer than 100 mesh it may be essential to good work.[1]

APPLICATION AND RESULTS

In respect to tonnage, the greatest use of magnetic separation is in the concentration of low-grade magnetite ores. Such use is made at Mineville on the eastern slopes of the Adirondacks, New York; in New Jersey; at Moose Mountain, Ontario; in the Eastern Mesabi district, Minnesota, and elsewhere in northern America; and in northern Norway, in Sweden, in Finland. Roughly about two million tons of ore are treated annually in each of the three principal districts, Mineville, Norway, and Sweden.

When purely concentrating these ores, the ratio of concentration is generally of the order of 2 : 1. But when magnetic treatment also breaks an undesirable association with apatite, pyrite, etc., the ratio of concentration may be lower and yet be associated with an abundant recompense; apatite so separated may indeed be a source of revenue as a fertilizer, as at Mineville, New York, in Sweden, and elsewhere.

Some magnetite ores are Bessemer ores both in the crude and in the concentrate; others will yield a Bessemer concentrate by any magnetic separation; while others, again, require finer grinding and wet separation for the production of this material. Non-Bessemer concentrate is common; treated in the basic-lined converter such concentrate gives up its phosphorus, the resultant slag being valuable as a fertilizer. All these different classes of ore may exist in the same field.

At Mineville, typical results of magnetic treatment of material from the 'Old Bed' show an increase of the iron from 59 per cent to 67 per cent and a concomitant decrease of the phosphorus from 1·7 to 0·6 per cent. At the Witherbee-Sherman No. 4 Mill, treating the leanest ore in the district, the iron content is increased from 32 to 64 per cent while the phosphorus is decreased from 0·1 to 0·025 per cent (Fig. 383).[2] This ore as it comes to the mill consists of clean magnetite, lean milling ore, quartzose rock, quartz, and hornblende gneiss. On broad lines, the treatment given consists of breaking to 1½ in., drying the fines, close-sizing, removal of clean magnetite on the one hand and waste rock on the other, as early

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[1] E. W. Davis, 'The Magnetic Concentration of Iron Ore,' *Bulletin of the University of Minnesota*, Vol. XXIV., 1921; or *Bulletin No. 9, Minnesota School of Mines Experiment Station.*

[2] Pellett, *E. & M.J.*, March 14, 1914, p. 549.

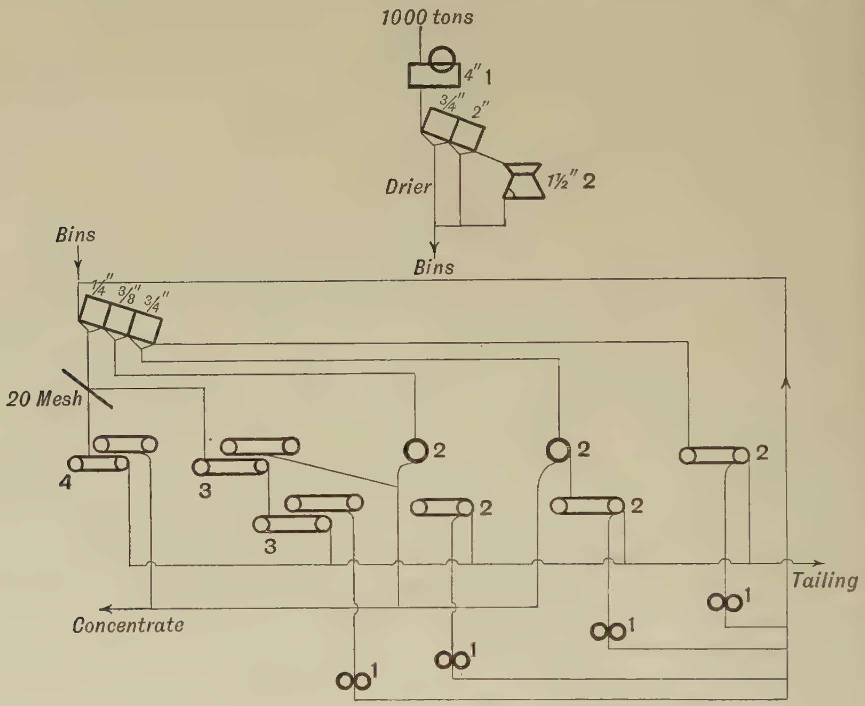

FIG. 383.

Magnetic Concentration at Mineville, New York.—Flow-sheet. The material after crushing to 1½"—2" is divided into five sizes. Of these sizes only the first and the last are treated by separators in parallel, the coarse material being divided into a middling and a tailing, the fine material into a concentrate and a tailing. All the three intermediate sizes are treated by separators in series, the first separator of each series making a concentrate, the second making a middling. All the middlings after recrushing by rolls are returned to the head of the circuit. Drum separators are used for coarse cobbing, pulley separators to discard a worthless tailing, and belt machines to treat the fine material. The block figures indicate the numbers of the different machines employed to treat 1000 tons daily (pp. 527, 533, 553, 656).

as possible, by drum, pulley, and belt separators, in succession; further crushing by rolls and retreatment of the middlings, the final crushing being to about 20 mesh. The resultant tailing contains 6 per cent of iron, the recovery is about 90 per cent, and the milling cost about 12d. per ton, when treating about 1000 tons per day.

At Mount Hope, New Jersey,[1] the ore treated in a mill having a capacity of about 500 tons per shift of eight hours, contains about 45 per cent of iron, while the concentrate produced contains 60 per cent, and the tailing discarded 10 per cent. The ore is dried in a tower, broken, crushed, and screened, then treated successively on drum, pulley, and belt separators, material finer than 8 mesh not being re-crushed.

At the Replogle mine, near Wharton, New Jersey,[2] working a magnetite lens in gneiss, magnetic separation alone was unsatisfactory because of the martite present, for the recovery of which water separation was added. At another mine such martite was recovered by dry-gravity, that is, pneumatic separation.

To-day an endeavour is being made to treat magnetically the low-grade magnetites of the Eastern Mesabi, Minnesota, containing but 25 per cent of iron. This ore is broken to 3 in. when worthless material is discarded; then broken to 2 in. to make a second discard, and so on down to $\frac{1}{4}$ in., after which practically only half the original weight of ore remains. This $\frac{1}{4}$ in. material is then ground to about 100 mesh in ball-mills, the product being treated in magnetic log-washers. From these washers the concentrate is dewatered by Oliver filters, then sintered, the fines being subsequently removed by screening; sintering, while essential to the production of material in the proper physical condition for the blast furnace, also removes much of the sulphur from any remaining pyrite. This treatment, while successful technically, has not yet become established commercially, the total mining and milling cost being about $2 per ton of ore milled instead of about $1, as was hoped; had it been so, there would have been reasonable hope that the comparatively soft haematite containing 35 per cent of iron, of which haematite there is an abundance, would reward a roast-magnetic treatment.

At Sydvaranger, Norway, the ore, which contains 35 per cent of iron, is broken and then crushed to 20 mesh in ball-tubemills to be treated in Gröndal wet-separators for a concentrate containing 56 per cent of iron. Further grinding and retreatment of this concentrate brings the iron content to about 69 per cent, when it is briquetted. At Herrang crushing is taken to 2 mm., and at Pitkaranta in Finland to slime, separation being made in wet separators.

At Dunderland the large proportion of haematite precluded wet magnetic separation; an attempt was accordingly made to crush and concentrate dry, but the fine crushing found to be necessary was productive of so much dust that work was eventually stopped.

In the Rhineland and Westphalia, magnetic separation is employed to separate, after roasting, siderite from its gangue. The roasted raw ore is crushed, and then treated generally in two sizes, say, $\frac{5}{8}$ in.—$\frac{1}{4}$ in., and $\frac{1}{4}$ in.—0 in. by magnetic separators often of the Ullrich type. The roasted ore generally contains about 36 per cent of iron and 6 per cent of manganese, the corresponding figures for the concentrate being 50 per cent and 10 per cent respectively; this concentrate contains approximately 90 per cent of the iron and manganese originally in the roasted ore. In making the separation a middling is produced which undergoes further reduction in rolls and retreatment on the separator. The cost of this magnetic treatment is about 10d. per ton; that of roasting and crushing is additional.

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[1] Shapira, *E. & M.J.*, March 27, 1915, p. 559.

[2] Hubbell, *E. & M.J.*, October 2, 1920, p. 658.

However important magnetic concentration may be in the beneficiation of iron ores, it is perhaps more important in connection with zinc ores, that is, in separating pyrite and marcasite on the one hand and siderite on the other, from blende concentrate.

In the Upper Mississippi zinc-field, a roast-magnetic treatment is the adopted means of removing marcasite and pyrite from blende concentrate, such treatment being generally given at custom plants. At Wisconsin,[1] this concentrate, which in greatest part is a water concentrate, contains about 35 per cent of zinc and 20 per cent of iron, both in combination with sulphur. This concentrate, after lightly roasting and cooling, is magnetically treated, whereby the zinc content is first raised to 55 per cent of zinc, and then to 60 per cent, or higher; the magnetics from the first treatment contain sufficient sulphur to make them of value for the production of sulphuric acid, while those from the second treatment are returned to the roaster. The cost of this roast-magnetic treatment is approximately:

	S.	D.	
Roasting and separation .	6	0	per ton of green concentrate.
Receiving and shipping .	4	6	,, ,,
General	4	0	,, ,,
Total . . .	14	6	

In Sardinia, representative of European circumstance, the material treated is a blende-siderite middling from jigs. This middling, which contains about 30 per cent of zinc and 33 per cent of iron, is screened into three sizes, namely, 7—4 mm., 4—2, and 2—0 mm., each size being treated on its own separator. The resultant concentrate assays about 55 per cent zinc and 45 per cent iron.[2] At the Victoria mine, Westphalia, to avoid drying, a similar middling is treated by the Ullrich wet separator, no sizing being necessary as all the material is finer than 3 mm.

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[1] Deutman, *E. & M.J.*, June 28, 1919, p. 1107.

[2] C. W. Wright, *E. & M.J.*, December 4, 1915, p. 911.

Of special interest is the magnetic treatment at Franklin Furnace, New Jersey, where franklinite and tephroite are separated from zincite, willemite, and non-magnetic gangue. The ore there is broken to 0·5 in., dried in Edison towers, crushed to 60 mesh, and then separated on Wetherill machines. The magnetics, which contain iron, manganese, and zinc, are used for the production of zinc oxide and spiegeleisen, while the non-magnetics are jigged to remove calcite, etc., giving zinc concentrate suitable for the production of spelter.

Another exceptional application of magnetic separation to zinc ores is that at Hanover, New Mexico, where the ore is a hard dense mixture of blende, pyrite, and galena, with typical contact-minerals and limestone, all of these constituents, with the exception of the galena and limestone, being magnetic, more or less. There, after crushing, the sulphide ore is divided into eight different sizes between 10 and 150 mesh, the material still finer being removed by the dust separator, and then treated on Wetherill machines. The resultant products appear far from clean.

Before the advent of flotation, magnetic separation was also applied to the treatment of zinc middlings at Broken Hill, New South Wales, these middlings containing rhodonite, garnet, blende, etc. Results, however, were far from satisfactory; the rhodonite and garnet were roughly separated as the most magnetic product; then the blende as a less magnetic product; and finally the remaining sulphides and gangue as a mixed non-magnetic product.

Magnetic separation has also rendered invaluable service in recovering wolframite and other valuable minerals from tin concentrate, the cassiterite becoming cleaned in the process. In Cornwall, where the ore is crushed by stamps to 12—30 mesh, the concentrates obtained by water treatment are roughly in three sizes, known respectively as coarse crop, fine crop, and slime. These three sizes are roasted separately, or at least the ‘ crop ’ tin separately from the slime tin, most of this latter being finer than 200 mesh. Associated with the cassiterite and wolframite there is always some pyrite, mispickel, and a little chalcopyrite. Roasting is usually continued till practically all the arsenic and sulphur have been driven off, though abundant air is excluded that the iron may not be entirely converted to ferric oxide, which, passing into the same magnetic product, would cause a contamination of the wolframite.

Upon this roasted material Wetherill separators are used, machines with two pairs of magnets for the crop tin, and others with three pairs for the slime tin. Treating the crop tin, the first magnets with 6 ampères of current remove the bulk of the iron oxide, while the second magnets using 15 ampères remove at the first field a mixed iron-wolframite product and at the second a fairly clean wolframite, the cassiterite passing on. The iron oxide so removed usually contains a little tin and wolfram, to recover which it is reground and water-treated again. The wolframite may likewise be contaminated with iron and contain some entangled tin; to remove such iron and recover such tin, this product is pickled for a number of days in a hot sulphuric-acid solution of about 15 per cent strength, after which it is dried and passed again over the separator; any copper in the spent liquor is precipitated on iron.

With slime tin, if there be three pairs of magnets these will be energized with about 6, 12, and 18 ampères respectively, or if two pairs with 6 and 18 ampères respectively; it is the experience in this separation that fine material requires stronger current. The necessary adjustment of the air-gap is readily obtained by placing shims upon the flat poles under the belt.

The capacities of the Wetherill separators in Cornwall vary from 3 to 8 tons per day of 24 hours, according to the size of the machine and the grain of the material.[1]

At the Launceston Works, Tasmania,[2] the concentrate treated magnetically contains cassiterite, bismuthinite, wolframite, together with pyrite, magnetite, and pyrrhotite. Of this material four sizes are made from $\frac{1}{8}$ in. downwards, each size being treated separately on a four-field Wetherill machine with 20 ampères available. The magnetite and much of the pyrrhotite are removed at the first and second fields, the wolframite at the third and fourth, leaving the other minerals in the non-magnetic product. This last product is roasted to render the pyrite magnetic and then retreated to remove the iron; the wolframite, when contaminated with pyrite, receives a similar but separate treatment. Eventually the clean wolframite assays about 70 per cent WO_3, while the cassiterite-bismuthinite product is likewise clean enough to market.

Again, at Ancia, Bolivia, wolframite is separated from a middling containing cassiterite, pyrrhotite, pyrite, etc., by Wetherill separators, 4 ampères being used on the first pair of magnets and 12 ampères on the second. This middling, of size 8 mm. downwards, is first ground to 2 mm., and then treated apparently without sizing. The first magnet separates pyrrhotite, which is ground again to release 5 per cent of contained tin; the second magnet moves material containing about 50 per cent of WO_3 and 20 per cent of iron sulphide. This second product is roasted to render the pyrite removable by a separate treatment using only 2 ampères, the second magnet then making a shipping concentrate assaying about 60 per cent WO_3.

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[1] Dietszch, *Trans. I.M.M.* Vol. XV., 1905, p. 10.

[2] Hitchcock and Pound, *Trans. Aust. I.M.E. Abst. M. & S.P.*, March 13, 1920.

The magnetic treatment of a tin concentrate or middling is not only of benefit when there is marketable wolframite to recover, but also when there is fine pyrite to separate from fine cassiterite, much tedious treatment and retreatment by water being thereby avoided. At Llallagua, for instance, from which district a substantial proportion of the Bolivian tin comes, dead-roasting and subsequent buddling of the pyrite-cassiterite middling has given place to magnetic roasting and magnetic separation. This middling, the particles of which are about 1 mm. in size, assays about 15 per cent of tin and contains about 27 per cent of sulphur in the form of pyrite. With such an amount of sulphur little or no fuel is required to roast to the magnetic sulphide, the burning of just over half the sulphur present being sufficient to keep the temperature above the ignition temperature of pyrite. The roasted material, lustrous, black, and containing 10—12 per cent of sulphur, is borne by water to Stern separators which separate a magnetic portion still containing about 22 per cent of sulphur, and a non-magnetic portion containing but 1—2 per cent, the former being reground and tabled, the latter being relatively-clean tin concentrate.[1]

Similarly, magnetic separation has been employed with great advantage in the Malay States to clean alluvial tin concentrate contaminated with ilmenite ; and in Nigeria to remove ilmenite, columbite, monazite, etc.

Another application of magnetic separation, illustrating its wide use, is that at the scheelite-gold mines near Otago, New Zealand. There the gold is largely recovered by amalgamation and cyanidation. But between these two treatments a scheelite-pyrite concentrate is made by tables, which, being roasted, is magnetically separated into a magnetic product containing about 5 oz. of gold per ton, and a non-magnetic scheelite which after retreatment by water is clean enough for the market.

Magnetic separation has also been of signal service in the recovery of monazite from the resistant minerals found with it in beach deposits. Such monazite sands on the shores of Southern India and Ceylon contain about 3 per cent of monazite. Brought by water concentration on tables or otherwise to about 12 per cent, these sands are treated on magnetic separators, Rapid or Wetherill, the accompanying ilmenite being separated in the weaker fields, the yellow and valuable monazite in the stronger fields, leaving a non-magnetic product consisting largely of zircon and quartz. A similar procedure is followed in Brazil.

Finally, but without exhausting the possible applications, magnetic separation is sometimes applied to the recovery of magnesite. Roasted magnesite is often magnetic by reason of contained iron, while the calcareous and argillaceous gangue is non-magnetic.

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[1] Copeland and Hollister, *E. & M.J.*, 1915, September 25, p. 573.

CHAPTER XII

ELECTROSTATIC, PNEUMATIC, AND CENTRIFUGAL SEPARATIONS

Electrostatic Separation

If a particle be brought into contact with an electrically-charged body, the charge will diffuse over the surface and through the substance, until finally it will reside wholly upon the surface and the particle and the body will be at the same potential. This diffusion may be instantaneous or it may take time, even days. During such time the particle is subjected to the inductive influence of the charged body and, at first, is attracted; but this attraction is gradually annulled by an increasing repulsion, which finally prevails, and the particle is repelled. Each particle has a specific time-element in which, under given conditions, it reaches a given potential, this time-element depending upon its conductivity and capacity. The conductivity of particles depends chiefly upon the nature of the surface; the capacity depends upon the size and shape of the particle, and also upon the character and number of the neighbouring particles.

In particular, if a pith ball suspended by a conducting thread be brought near a body charged with electricity, it is attracted to that body; such a system constitutes the simplest of electroscopes. With contact made, if the body be a conductor, its charge is immediately discharged and the ball falls away; but if it be a non-conductor, immediate discharge takes place only at the point of contact, the remaining charge being still sufficient to hold the ball. Or, if the pith ball be suspended by a non-conducting thread, it is first attracted and then after contact sharply repelled, the ball becoming charged.

In these phenomena, the pith ball, by reason of atmospheric moisture absorbed on its exterior, is a conductor, that is to say, it readily conducts electricity. Relatively to conductors, materials described as non-conductors are in reality bad conductors, there being no non-conductors in an absolute sense and irrespective of time.

Nearer to the subject of ore-dressing, if a mixed mass of conducting and non-conducting mineral particles, placed upon a conductor, be approached by a charged body, the conducting particles pass on to the conductor their induced charge of the same sign, and with their charge of opposite sign leap upwards toward the charged body. In their upward movement they reach the body, when if that body be a conductor they become charged and are repelled, whereas if it be a non-conductor and not excessively charged they are held.[1] The non-conducting particles, on the other hand, taking time to become polarized make no movement.

Such differences in attraction and repulsion, recognizable as the effects of specific conductivities, determine the different paths taken by conductors and non-conductors in electrostatic separation.

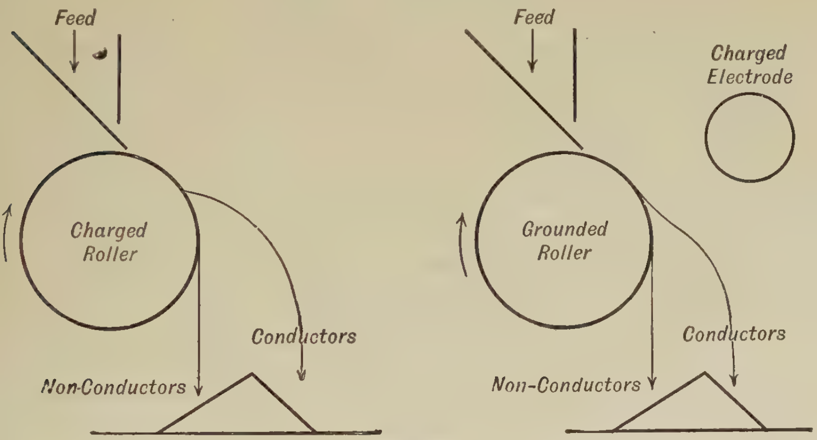

Fig. 384.

Electrostatic Separation.—Diagrams. That to the left represents separation by repulsion upon contact with a charged roller, as originally suggested by Blake-Morscher. That to the right represents separation by attraction from a grounded roller towards a charged electrode, as applied in the Huff Separator (pp. 561, 563).

If a mass of mixed particles be fed on to a revolving cylinder charged to a high potential the conducting particles become charged and repelled, while the non-conductors fall undeflected. This principle was first applied commercially in the separation of minerals by Blake and Morscher at Denver, Colorado, in 1901, under potentials at first as high as 250,000 volts, but afterwards as low as 20,000 volts (Fig. 384).

Again, if a mass of particles sliding over a grounded conductor passes into the field of a charged electrode, the conducting particles, being attracted by that electrode, are deflected from the path in which the non-conducting particles continue to move under their momentum and gravity. This method of separation was first applied on a commercial scale by the

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[1] Crook, *Mineralogical Mag.*, 1909, vol. xv. p. 260.

Huff Electrostatic Company of Boston, Massachusetts, in 1908, using 10,000—25,000 volts (Fig. 384).[1]

In general, under such treatment, opaque minerals with metallic lustre declare themselves good conductors; those with submetallic lustre, moderate conductors; while clear minerals with vitreous lustre are bad conductors. In particular, important minerals group themselves as follows:

Good Conductors.	Moderate Conductors.	Bad Conductors.
Chalcopyrite	Wolframite	Calcite
Pyrite	Cassiterite	Quartz
Galena	Sphalerite, etc.	Fluorite
Molybdenite		Garnet
Magnetite		Monazite
Ilmenite		Most silicates, carbonates, sulphates, etc.
Graphite		
Most sulphides, etc.		

Impure minerals behave irregularly; blende containing much iron or manganese may, for instance, be a relatively good conductor. The condition of the surface may also greatly alter the conductivity; if there be condensed moisture upon the surface, non-conductors will behave as conductors, for which reason electrostatic separation is only possible when the ore is dry, and is best achieved when the ore is warm enough to keep atmospheric moisture away. Or, if the surface be covered with an incipient deposit of a conductor, it will behave as such; a weak solution of copper sulphate will, for instance, quickly deposit such a film of copper sulphide upon blende, that blende can be separated as a conductor from, say, barite, a non-conductor. Apparently, also, heat tends to increase the conductivity of some carbonates.

In view of these facts, the possibility of applying electrostatic separation to any given ore in any given atmosphere, must be made the subject of special test.

Though conductivity in respect to dynamic electricity may have no absolute pertinence in respect to static electricity, the following figures of relative conductivity are not misleading in their order:

RELATIVE CONDUCTIVITIES

Silver	681,000	Chalcocite	91·0
Copper	634,000	Pyrite	41·7
Gold	455,000	Magnetite	1·24
Iron	113,000	Chalcopyrite	0·983
Covellite	8,000	Cuprite	0·025
Galena	3,350	Siderite	0·00014
Graphite	700	Quartz	$0{\cdot}84+10^{-14}$
Pyrrhotite	119		

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[1] Wentworth, *Trans. A.I.M.E.*, 1912, Vol. XLIII. p. 411.

Greater conductivity means greater static charge in a given time, whether induced or communicated by contact. Separation demands such forces of repulsion or attraction as shall sufficiently deflect the conducting particle from its path. These forces are between electric charges; electric charges reside on the particle surface, and are proportional to the extent of surface. Deflection from the path requires the exercise of force proportional to the mass, that is, to the volume. Accordingly, larger particles having relatively little surface in respect to mass, are deflected less than smaller particles; and for effective separation there must be close sizing. Close sizing also avoids the entrainment of fine particles by attachment to, or by the bombardment of, the coarse particles. Practice sanctions the division of the material into the sizes: 8 to 12 mesh, 12 to 20 mesh, 20 to 50 mesh, and the material finer than 50 mesh, after elimination of the dust. Material larger than 8 mesh requires greater forces of deflection than provided by the ordinary design of machine.

Electrostatic Separators

The Blake-Morscher separator employed a roller statically charged to a high potential by a frictional machine, generally a mica-plate generator. The framework of the machine was of wood, and not electrically grounded. Fed on top of the roller, which was a dozen feet in length and about 6 in. diameter, the conductors were repelled, while the non-conductors either fell right away or were carried round by the roller till scraped off (Fig. 384). The material treated was usually a water concentrate from zinc ore, this concentrate containing blende and marcasite principally, say, 35 per cent of zinc and 20 per cent of iron. Treating such material, the zinc content would be raised to 50 per cent and the iron reduced to about 7 per cent. Though such a result would be satisfactory if regularly obtained, it was not possible to avoid fluctuations of the potential and irregular work; not only were dust and moisture fatal to the proper functioning of the generator, but fluctuations also occurred even when these were eliminated. In consequence the use of this separator, instead of extending, was eventually given up.

Huff Separator.—This separator employs a grounded roller for fine material and grounded plates for such coarser sizes as require no roller to maintain a steady stream; the supports for these parts are of metal. Facing them are insulated electrodes charged to the requisite potential, 10,000—25,000 volts, by the use of an ordinary electromagnetic generator and transformer. Alternate current being generated, a rectifier is used in order that the electrodes may receive only a positive charge. Fed on to the grounded roller or plate, which, about 6 ft. in length and of small transverse dimensions, constitutes the neutral electrode, the conducting particles are attracted towards the positive electrode; falling by the way, they nevertheless are deflected from the path along which the non-conducting particles continue (Fig. 385). In another design, used principally to separate graphite and mica, the material falls first on to a negatively-charged plate, whence the conductors are repelled, to be attracted and then repelled again, by a positively-charged plate which the non-conducting particles never reach.

Whatever the design, no single passage between electrodes suffices to disentangle the particles of even the shallowest stream, and in general there are five or six of such fields in a falling series (Fig. 386). In turn, the two products of such a series become the respective feeds to two finishing series below, whence finally three products emerge, non-conducting particles, conducting particles, and middling.

Compared with the Blake-Morscher machine, the Huff separator has the advantage of a generator giving a steady potential and easy to control. A 3-h.p. motor is capable of generating sufficient charge for a number of these machines. Being almost entirely of grounded metal, this separator is relatively free from electric disturbance and safe to operate. It is at the moment the only electrostatic separator available.

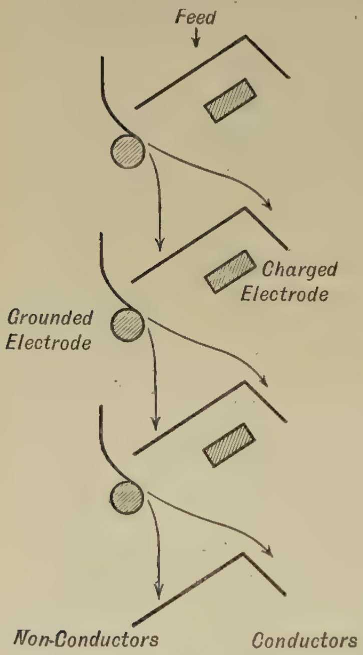


FIG. 385.

Huff Electrostatic Separator.—Diagram. The grounded electrode is shown as a fixed rod and not as a roller, this design being suitable to material which runs freely; this fixed rod is enveloped by a metal sheath which, extending above the roller, guides the falling stream. Three fields in series are shown, usually there are six. The normal design places three full series to make a complete machine, the first being a roughing series, producing two products which are fed respectively to two finishing series. Of these two finishing series, one makes a separation between non-conducting particles and middlings, while the other separates middlings and conducting particles (p. 563).

Practice and Results

As already indicated, electrostatic separation requires the material to be perfectly dry, sometimes, indeed, to be warm and dry; accordingly, an adequate drier, say of the cylindrical type 20 ft. long by 4 ft. diameter, is generally included in a typical plant. It also requires the material to be closely sized, say into four sizes, 8 to 12 mesh, 12 to 20 mesh, 20 to 50 mesh, and below 50 mesh. The material treated being generally a concentrate obtained by water does not, even at the outset, contain much impalpable material, but the amount of such material is further reduced by a suction fan exhausting through the drier. The finest size will nevertheless contain much material which will pass 120 mesh, some even passing 200 mesh; where there is relatively much of such fine material, the dust around the machine will be drawn away by an exhaust fan, sometimes to be collected in a bag-house.

The Huff separator was first used in 1908 in Wisconsin, to separate marcasite from a blende concentrate, the material treated containing about 28 per cent of zinc and 24 per cent of iron, and the resulting product about 48 per cent of zinc and 4 per cent of iron.

In 1909 at Midvale, Utah, it was used to treat a complex concentrate containing zinc, lead, copper, gold, and silver, from which it made a zinc product and a lead-smelting product, both of which were marketable.

In turn, at Cananea, Mexico, it treated a zinc-copper middling assaying 7 per cent copper, 15 per cent iron, and 30 per cent zinc, making a zinc product containing 55 per cent and 5 per cent iron, and a copper product containing 15 per cent copper and 10 per cent zinc, in addition to iron.

At Ouray, Colorado, it separated blende from heavy gangue contaminating a water concentrate, after immersion of this concentrate in a 0·5 per cent copper-sulphate solution for about fifteen minutes, the consumption of copper sulphate being about 1·5 lb. per ton. By this treatment the zinc content was raised from 40 per cent to 50 per cent.

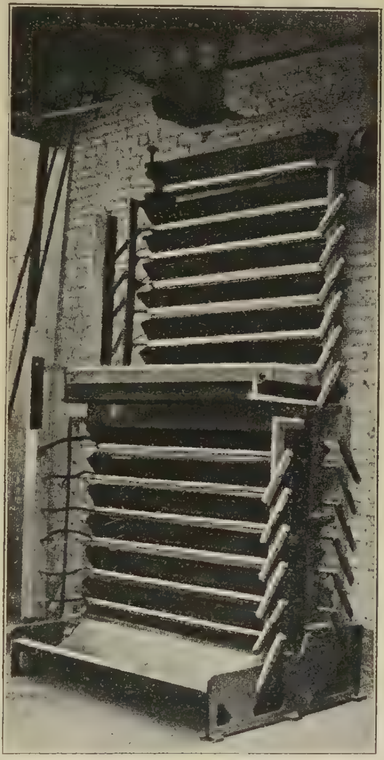


FIG. 386.

Huff Electrostatic Separator.—General View. A rougher series serving two finishing series below, the final products being mineral, middling, and tailing. This machine weighs about a ton and is stated to have a capacity of about 5 tons per day (p. 564).

Tests made recently at Rolla, Minnesota, upon a blende-fluorite ore from southern Illinois, such ore containing 7 per cent lead, 28 per cent zinc, 36 per cent fluorite, 10 per cent silica, and 5 per cent calcite, showed that, after crushing to 10 mesh and removing the galena and light gangue by water and treating the blende-fluorite middling with copper sulphate, it was possible to obtain electrostatically a marketable blende and a marketable fluorite. The recovery of blende was, however, not nearly so high as could be obtained by flotation after grinding to 65 mesh; on the other hand, the recovery of the fluorite was excellent, and the product cleaner and coarser than was possible by flotation.

In Australia the Huff process was tried upon a molybdenite ore.

Compared with magnetic separation, electrostatic separation avoids a preliminary roast, and by retaining the sulphur, delivers an iron product valuable for sulphuric-acid manufacture; where also galena is present with blende these two are separated. Nevertheless, the roast-magnetic treatment, because of the cleaner zinc-product it makes, and because of its more certain operation, prevails. Even at its best, electrostatic separation makes relatively much middling. Such middling is returned for retreatment, beginning at the drier.

It seems probable that the most fruitful field for electrostatic separation will be the removal of mica from graphite.

DIELECTRIC SEPARATION

Electrostatic separation may also be accomplished by an appeal to the dielectric properties of minerals. A process based on these properties is being developed by Dr. H. S. Hatfield, London, from whom much information concerning it was obtained.

A dielectric is a medium which, though non-conducting or insulating, is capable of propagating electric inductive forces; the degree in which it accomplishes this propagation, measured in respect to propagation through air as unity, constitutes the specific dielectric property, or, in other words, the ‘dielectric constant’ of the medium.

Dielectric properties come under notice particularly in electric condensers such as the Leyden jar, the elements of which are two conducting armatures separated by a dielectric medium, glass, for instance. The capacity of such a condenser will depend upon the dielectric property of the medium employed, and the ratio of the capacity with a given medium to the capacity with air, is known as the ‘specific inductive capacity’ of the substance employed as medium, that is to say:

$$K \text{ (\textit{specific inductive capacity})} = \frac{C_1 \text{ (\textit{capacity with medium})}}{C \text{ (\textit{capacity with air})}}.$$

Watson (*Text-book of Physics*, 4th ed. p. 639) gives the following values for K:

Air	1·0	Shellac	3·3
Turpentine	2·2	Glass	6·0
Benzene	2·3	Mica	8·0
Ebonite	2·5	Alcohol	25·0
Sulphur	3·0	Water	76·0
Petroleum	3·1		

The specific inductive capacity is a particular range of the more general dielectric constant, but generally the two expressions are used synonymously. The latter may be determined by the time taken to discharge an electroscope, or by the Wheatstone bridge; for quartz it is about 5, for clean cassiterite about 20—30, and for cassiterite contaminated with iron, about 50; conductors, such as pyrite, galena, etc., behave as if their dielectric constant were infinitely great. Of liquids, the constant for kerosene is about 2—3; those of aniline, nitro-benzene, about 36, while those of amyl-alcohol, acetylene tetrachloride, etc., are similarly high. The low constant of kerosene may be stepped up even to 36 in the end, by successive small additions of nitro-benzene, this latter material even in small quantities profoundly modifying the dielectric properties of kerosene; similar stepping-up of other mediums with low constants may doubtless be made.

This method of dielectric separation consists in suspending the material in an insulating liquid medium, the dielectric constant of which lies between those of the two minerals to be separated. So suspended, the material falls between electrodes connected to an alternating supply, whereupon those particles of higher dielectric constant than the liquid are attracted to the electrodes while those of lower constant are repelled. Alternating current is used because with direct current the ultra-fine particles by reason of their own electric charges would move towards and attach themselves to one electrode or the other, that is to say, there would be a display of cataphoresis; the larger particles, on the other hand, would be attracted to the electrodes to receive a charge and be repelled.

Ordinary electrostatic separation depends upon the particles receiving effective charges and being repelled, high voltages are therefore necessary. In dielectric separation the force of attraction or repulsion depends rather upon the rate at which the strength of the field varies; accordingly, an ordinary voltage of about 200—250 suffices, the requisite strength of field being obtained by bringing the electrodes very close together, say to within about $\frac{1}{6}$ in., and the requisite divergence by using fine parallel wires as electrodes.

In such a static field the force of attraction or repulsion, as the case might be, is given by the formula

$$F = \frac{3K_1(K_2 - K_1)}{8\pi(K_2 + 2K_1)} V \times \frac{dF^2}{ds},$$

where K_1 *is the dielectric constant of the liquid,*
,, K_2 ,, ,, ,, *solid,*
,, F ,, *strength of the electric field,*
,, s ,, *distance from the electrode.*

For a given substance and a given liquid this formula becomes

$$F = KV\frac{dF^2}{ds},$$

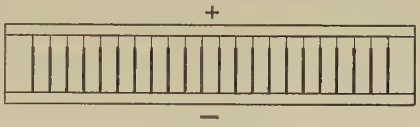
which it is seen is identical to that pertaining to a magnetic field (p. 523). There is, however, this difference between the two fields, that whereas in the magnetic field K, which is the magnetic susceptibility, varies extremely, in the electric field it is a function involving a subtraction, and though the range of dielectric constants is infinite the range of attractive force is very narrow ; if, for instance, the dielectric constant of the liquid be 2, and three minerals have constants of 5, 30, and infinity, respectively, the values of K will be 0·08, 0·19, and 0·25, respectively. Conducting particles accordingly move with much the same force as poor conductors. A difficulty with conducting particles is, however, that these may cause sparking and consequent rupture of the electrodes ; but this can be avoided by insulating one set of the electrodes. In this connection it should be remembered that though the generated voltage be only 200—250, the voltage in the intense field between the electrodes will be very much higher and of the order of 1000—2000 volts.

Arrived at the electrodes the particles of high constant attach themselves, others filing behind till the gap between the electrodes is bridged, and the field is loaded ; the attached mineral may then be dropped by breaking the circuit. The particles of low constant have in the meantime settled freely to the bottom, whence they have passed out.

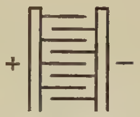
Though the medium be non-conducting, and in consequence the current passing be little, there is, when the frequency of the current is low, some evidence of an ‘ electric wind ’ such as might keep the very fine particles from attachment. This effect is not sufficiently pronounced at ordinary frequencies of 50—60 periods to be detrimental, while at very high frequencies it ceases.

Other than this, there are no forces opposing separation, and the material, even the finest, moves with precision, the mineral one way, the gangue the other, a difference of one unit in the dielectric constant being sufficient to make a pronounced separation. This method of separation should accordingly be suited to the recovery of slime mineral.

It has, however, only been applied in the laboratory and on a laboratory scale. The liquid medium used has been variously aniline, nitro-benzene, amyl-alcohol, acetylene-tetrachloride, and kerosene contaminated with nitro-benzene. Aniline was used because, while non-volatile at ordinary temperatures, it could be subsequently recovered by heating the products, but it is very expensive; acetylene-tetrachloride is poisonous and expensive; kerosene contaminated with nitro-benzene has the advantage of being much less-conducting than aniline, and much cheaper. Probably the best medium would be some volatile liquid requiring little heat for its eventual recovery; the operation would, however, then require to be conducted in a closed chamber. To ensure complete freedom of movement for the particles in the necessarily non-aqueous suspension an appropriate deflocculent, oleic acid, neatsfoot oil, etc., must be added.



Plan



Cross Section

Fig. 387.

Hatfield Dielectric Separator.—Diagram. The design illustrated is that of a cell for a discontinuous machine; when such a cell has become loaded, the mineral is detached by breaking the circuit, and collected by flushing with water. The gangue passes continually away during the period of loading (p. 569).

A cell in which this separation could be effected has for its sides two insulated plates set about 2 in. apart, each plate having rows of projecting pins upon it, the rows upon one plate fitting in between those upon the other. With these two plates connected respectively to the two terminals of a circuit, and with the space between them filled with the non-conducting medium, electrostatic fields would exist between adjacent pins (Fig. 387). Feeding the powdered material on top, the particles of higher dielectric constant than the medium would cling to the pins, while those of lower constant would sink to the bottom, whence they could be withdrawn.

For continuous work a coiled electrode around a straight electrode, the two forming a unit electric pair, is suggested. The dimensions of this unit would be small enough that the pair when lifted out of the medium would, by surface tension, carry its fill of medium, to the maintenance of the electric field. Arrived over a proper place for discharge, the attached particles would then be dropped by breaking the circuit. Messrs. Sandycroft of Chester have in hand the making of a commercial machine on these lines.

For making small tests upon powdered ore, dielectric pliers have proved useful, these pliers consisting of two wires, attached at one end respectively to the two terminals of a circuit, and at the other end brought to approach one another (Fig. 388). These pliers are manipulated in the depression of a watch glass containing the medium and a portion of the mineral powder. Should there be present a conducting mineral such as pyrite, one wire is completely insulated with paper. Around the working end of such pliers the particles of high dielectric constant are seen to align themselves with the lines of force.

Tests made upon the finest portion of a crushed Cornish tin ore gave a recovery of 85 per cent of the cassiterite in an enriched but far from clean concentrate. Cobalt and nickel oxides, pitchblende, roscoelite, the last a complex vanadium mineral, have all been separated from gangue. Carnotite, an ore containing vanadium, uranium, and radium, can easily be separated after a slight chemical reduction.

This dielectric process would appear to have great possibilities with rare ores: the separation of vanadium minerals, for instance. In mineral separation it appears an entirely new application of the particular physical property. It differs on the one hand from ordinary electrostatic separation in that it is not operative by reason of charges acquired by contact; and on the other from ordinary 'cataphoresis,' in that this latter is a migration of colloidal particles towards an electrode by reason of charges they possess of themselves.

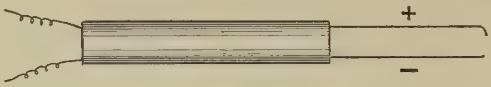


FIG. 388.

Dielectric Testing Pliers.—Diagram. Such pliers have been found very useful in making laboratory separations of minerals which could not be separated otherwise, and in testing the possibilities of dielectric separation with given ores. The electrostatic field is between the curved end of one wire and the straight end of another (p. 569).

The Sutton-Steele Company in the United States advertise a dielectric separator which operates on what they call the Dielectric Hysteretic Impedance in the particles. This machine possesses a number of rollers, and in appearance resembles the ordinary electrostatic separator. Working dry in air, its principles obviously differ from those of the process just described. No records of results obtained have been noticed.

PNEUMATIC SEPARATION

In discussing water concentration it was seen that the medium in which separation takes place is, by reason of the discriminating resistance it offers, a large factor in securing that differential movement between particles which eventually results in separation. So much is this the case that separation in water is also largely separation by water, and the process is deservedly described as water concentration. Similarly, when the medium in which, and largely by which, separation is effected is air, such separation is described as air- or pneumatic separation.

The resistance of the medium is responsible for mineral particles falling in water faster than those of gangue ; when no resistance exists, as in a vacuum, all things fall together. Reciprocally, it is also the measure of the drag exercised upon particles exposed to a stream of the medium. This resistance is proportional to the density of the medium ; accordingly, a dense medium favours separation and *vice versa*, witness the perfect separation when the medium is heavier than the gangue. That being so, pneumatic separation, functioning in and by a light and tenuous medium, is at a disadvantage ; wherefore it demands closer sizing than does water concentration (p. 255).

The resistance of the medium also determines the absolute terminal velocities of falling particles, or, reciprocally, the rising velocities of the medium necessary to suspend them ; where the medium is light, resistance is less and velocities are greater. Here again air separation is at a disadvantage ; even with a moderate range of size the air velocity appropriate for the coarse particles would project the fine particles beyond the confines of the appliance. Further, very fine material, say finer than 300 mesh, readily diffuses in air, defying separation and submitting only to undivided collection ; and, at the other end, particles larger than about 5 mesh require a greater volume of air than usually could be afforded.

Other disadvantages are : the necessity, if the ore be not dry of itself, to dry the ore ; the necessity to dry-crush the ore ; to dry-screen it ; and to mechanically convey the products, since a stream of air, unlike one of water, cannot be confined in launders.

A point decidedly in favour of pneumatic separation is that air keeps the bed of powdered ore loose, permitting the particles to move readily to their respective positions under density ; water, on the other hand, as it drains away, tends to compact the bed, cramping the necessary freedom of movement. Another favourable point is that the impalpable material unavoidably made in crushing, is more readily settled as dust than as slime from a water suspension. Air also is present everywhere and does not require to be conserved.

The balance of disadvantage is, however, such that pneumatic concentration is only used when water is not available, as in the arid regions of the United States, Mexico, Western Australia, South-west Africa, and elsewhere ; or when a light and flaky material, such as graphite, is to be separated; or where, in crushing, the valuable mineral, being friable, goes to dust while the gangue remains granular, as with some uranium and vanadium ores.

Pneumatic Separators

Pneumatic separation being, like that achieved by water, a density separation, the appliances used have many resemblances to those employed with water: jigs, inclined tables, and shaking tables, are all represented. In addition, there are types peculiar to air separation: blowers, which separate by the differential deflection of the falling particles; and some accessory appliances, such as de-dusters and dust collectors. Centrifugal separators are described under a separate heading.

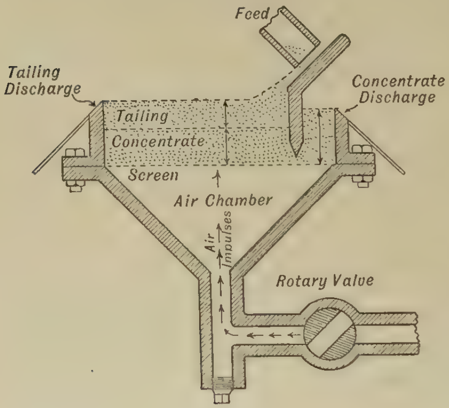


FIG. 389.

Plumb Pneumatic Jig.—Diagrammatic Cross-section (p. 572).

Plumb Pneumatic Jig.—In this jig, as with all jigs, there is an ore-box having a sieve for bottom; unlike ordinary jigs, however, this box is long and narrow, namely, 24—36 in. long and 3 in. in width (Fig. 389). Close to one side and along the whole length, a vertical gate extends downward to within a measured distance of the sieve; under this gate the concentrate creeps to be discharged over a lip running along this side of the box. The material to be treated is fed on the other side of this gate into the main portion of the box, the tailing being discharged over a lip running along that side. This discharge lip for the tailing is at a somewhat higher level than that for the concentrate, the two columns on either side of the gate balancing one another like two liquid columns, a heavy liquid within the gate, a lighter liquid without. Into the necessary mobile condition the particles are brought by the pulsation of air up through the sieve, pulsation being obtained by a rotary valve upon the air main. The number of these pulsations is about 400—500 per minute and their strength that of 10—30 lb. pressure at the valve, the higher pressure being for larger material. The ore-box sits on a hutch with sloping sides; along one side at the bottom of this hutch the air arrives through two entries; below this again are discharges through which from time to time any hutch-work may be removed. The whole body is of cast-iron, with steel and brass fittings (Fig. 390).

The material treated by these jigs is generally a rough water-concentrate or middling, containing but little very-fine material. For successful treatment such material requires to be closely sized, say between 12 mesh, 20 mesh, 50 mesh, and 120 mesh screens ; dust cannot be successfully treated ; nor material coarser than 5 mesh. Treating such material so prepared, the capacity is about 10 tons per day per square foot of sieve area for the coarser sizes, and less for the fine. The power consumption for air-compression is about 10 h.p. hour per ton treated.

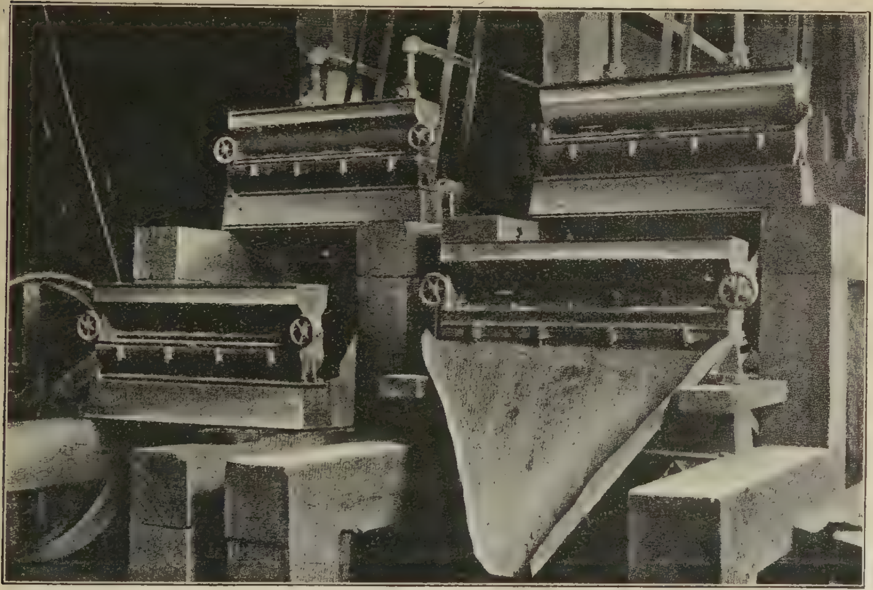


Fig. 390.

Plumb Jigs at the Sunnyside Mine, Colorado.—General View. The long length and narrow width of these jigs is noticeable. The hand-wheels at either end regulate the rate of feed, by moving the feed spout to widen or to narrow the feed slit. The double air-entry to the upper jigs is clearly seen (p. 572).

The pneumatic jig does not appear, however, to have a wide application. It has been tried at Bunker Hill, Idaho, upon lead ores ; at the Yellow Pine district, Arizona, upon zinc-lead ores ; in Alabama, upon graphite ; and elsewhere.

Sutton-Steele Pneumatic Table.—This appliance resembles the ordinary wet-concentrating table (Fig. 391). Its deck, which is 8 ft. long and

5 ft. wide, is smooth ; it has no riffles, but is covered with pervious cloth. This deck is mounted on a shallow wind-chest which receives air of 1—4 oz. pressure from a special blower. This chest in turn is supported upon a number of lath-like springs which lean over towards the head motion, after the manner of the Ferraris table. This head motion consists of a cam which operates a bell-crank lever to bring the table back, compressing at the same time a spring which then takes the table forward. This movement, which itself is differential, is made still more differential by the table support, which carries the table upward as it moves forward, and *vice versa.* From the wind-chest the air passing out by the pervious deck stratifies the ore fed at the back corner ; the gangue particles, forced to a riding position on top, roll down the table inclination to be quickly discharged along the lower edge ; the mineral particles, descending to the table surface, are carried forward by every impulse, and their arrival at the lower edge is delayed. Between these two positions any desired middling may be separated.

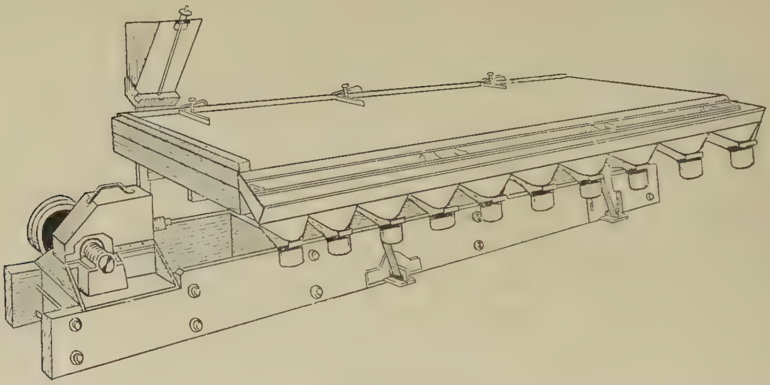


Fig. 391.

Sutton-Steele Pneumatic Table.—General View. The motion communicated to the table gives such a strong forward impulse that the material may be fed at the back corner, no long feed-box being necessary. The many pockets along the discharge indicate the many products which can be made. The table is seen to be supported, like the Ferraris table, on laths (p. 573).

The number of strokes is about 400 per minute, and the amplitude about one-quarter of an inch. Treating dry and closely-sized material the capacity of these tables is much the same as that of wet tables ; the separation of mineral from gangue is good. They have been used in the arid districts of the United States to separate galena from calcite and from blende ; in India, to separate monazite from shore sands ; and elsewhere.

Stebbins Pneumatic Table.—This table, which is about $9\frac{1}{2}$ ft. long by 3 ft.

wide, also resembles an ordinary water-table. Its deck is of sheet metal and is riffled, the riffles being about 1 in. apart. This deck is mounted on a shallow wind-chest served with a continuous blast. In the bed and between the riffles are slotted perforations equal to about 20—30 mesh. These perforations are placed with their length at an angle of about 45°

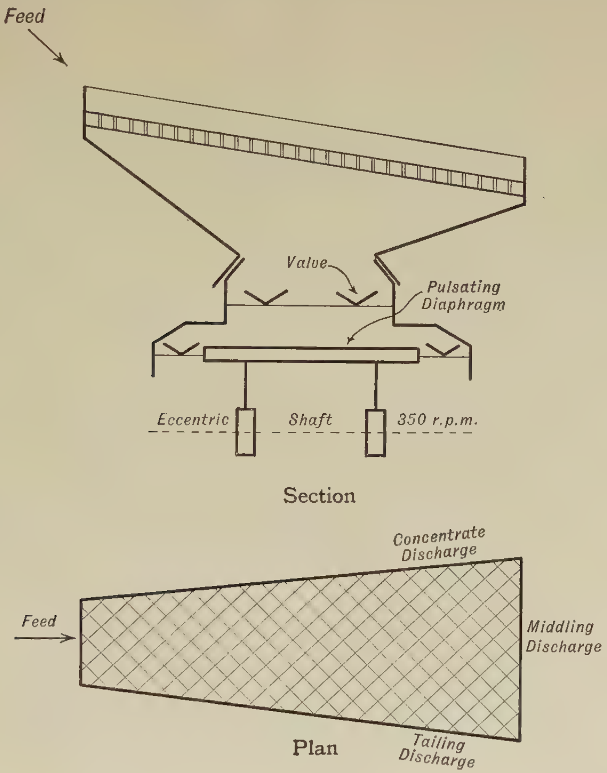


Fig. 392.

Hooper Pneumatic Table.—Diagrams. The section shows a universal joint around which the inclination of the table may be varied. The plan shows the cross slats of two frames, one superimposed upon the other. The slats of the lower frame direct the concentrate to its discharge, those of the upper frame performing the same service for the tailing (p. 576).

to the riffles; their form is such that the issuing air moves down the table, assisting thereby the discharge of the tailing. The concentrate moves forward with each table-movement, to be eventually discharged at the far end. This table has been used to treat placer gravel after removal of the boulders and coarse material; it has also been used to concentrate lead and lead-zinc ores. The self-contained blower with which it is provided requires about 6 h.p., the whole installation requiring about 10 h.p. For dust removal a hood and exhaust fan are provided.

Hooper Table.—This table is relatively long in the direction of the stream and of small dimension across (Fig. 392). Its deck consists of a fine grid over which a stout broadcloth is stretched; the slats of the grid are laid obliquely across the stream. This deck is supported on a wind-chest having a pulsating diaphragm for bottom, pulsation being accomplished at the rate of 350—400 per minute, by eccentrics upon a shaft beneath. Progression of the material down the table is the joint effect of an inclination of about 10° and the air pulsations. Under these pulsations the mineral sinks while the gangue comes to the top. To discharge these two products separately two frames are laid upon the deck, the lower one having its bars parallel to those of the grid beneath, the upper one having its bars crossing those of the grid; the lower frame directs the concentrate to one side of the table, while the upper frame directs the tailing to the other side. Treating sized material, this table has a capacity similar to that expected from a water table. It has been used in the various districts of the United States to separate galena, blende, etc.; and has satisfactorily separated corundum from quartz, the difference in specific gravity between these two minerals being only 1·3.

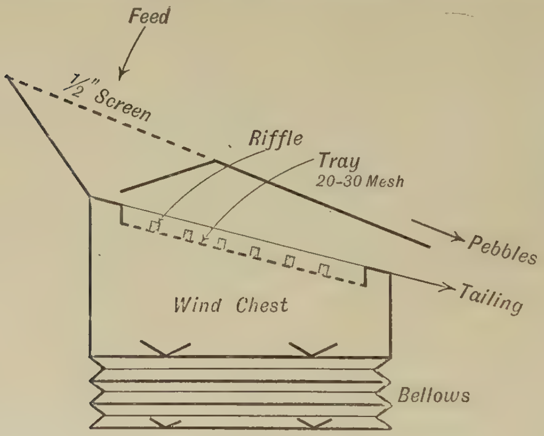


FIG. 393.

Dry Washer.—Diagram (p. 576).

Dry-Washer.—This is the general name for the various simple appliances, the design of which includes a porous inclined table up through which air is forced by a bellows, the inclination and the air pulsation effecting the progression and eventual discharge of the gangue, while the valuable mineral, generally gold, is caught by and collected behind riffles (Fig. 393). The inclined table takes the form of a tray 2—3 ft. long, about 18 in. wide, and a couple of inches deep, with a bottom of stout muslin or of thin metal-sheet pierced with holes about 0·05 in. diameter; spaced at regular intervals down the length are riffles. This tray is wedged firmly, but in a manner permitting ready removal, into the top of a wind-chest served either by a single or a double bellows. Above the tray is a screen with ½-in. holes to reject the coarser material, and a plate to guide the undersize back to the head of the tray. Sometimes to assist progression, the effort involved in working the bellows also imparts a side-to-side rocking motion to the tray. When it is judged that the space behind the riffles has become reasonably loaded with valuable concentrate, the tray is lifted and the concentrate is removed.

The dry-washer has been largely used to recover gold from auriferous gravel deposits in New Mexico, Arizona, Western Australia, and elsewhere where water is lacking (Fig. 394).

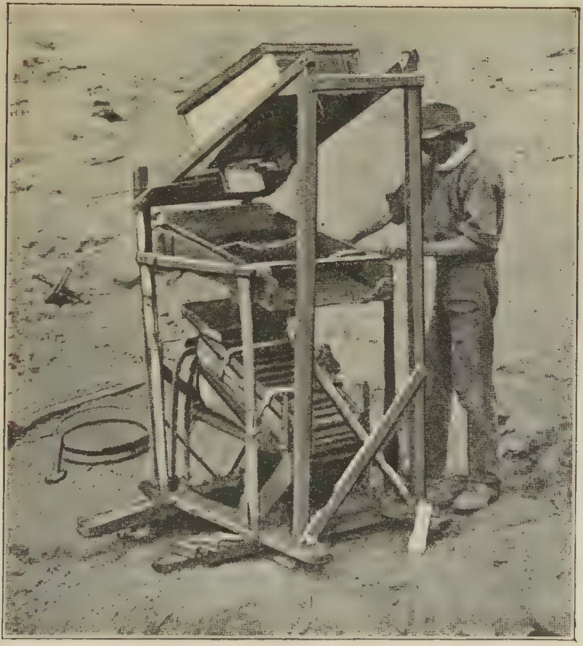


Fig. 394.

Dry-washer.—Washer in operation in Western Australia. On top is a hopper; midway is a screen; below is the washer proper. The riffles on the washer show plainly; the tray is seen mounted on top of ordinary bellows apparently worked from behind (p. 577). (*M. & S.P.*, August 21, 1915.)

Dry Blower.—So may be described all those appliances wherein a horizontal current of air causes the differential deflection of falling particles of ore, a greater deflection of the light particles, and a smaller deflection of the heavy; among such blowers, some suck the air across the falling stream, while others blow it across. The necessary current of air may be created by a centrifugal fan working in a casing or by a screw fan rotating in an enclosed opening (Figs. 395, 396). For the successful working of the appliance this current must be maintained uniformly across the plane of the falling ore; in addition, the ore must be completely dry and regularly fed in a thin stream, generally by a roller feeder.

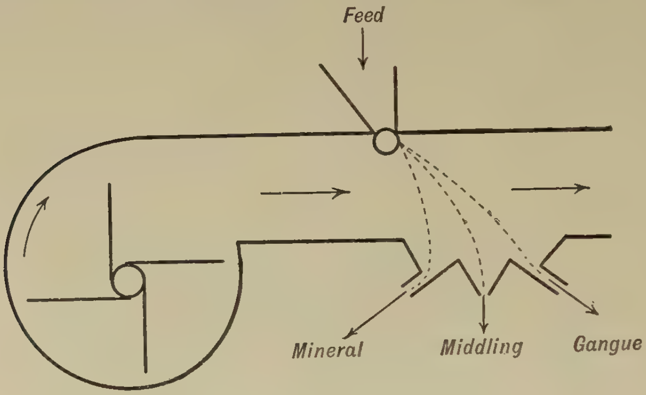


FIG. 395.

Dry-blower.—Diagram of the Blowing Type. Before the introduction of magnetic separation such a blower was sometimes used to clean alluvial tin-concentrate, blowing ilmenite, garnet, etc., away from cassiterite. The air-way is about a foot deep and across (p. 577).

Dry-blowers have now their principal use in the refining of graphite concentrate, this flaky and light mineral being deflected clear of the more vertical path taken by the equidimensional and somewhat heavier quartz-grains. In Australia and elsewhere dry-blowing for the recovery of alluvial gold is also achieved in a moderate breeze as the material is dropped from one washing pan to another; and in a similar way for the recovery of diamonds from the desert alluvials of South-west and South Africa.

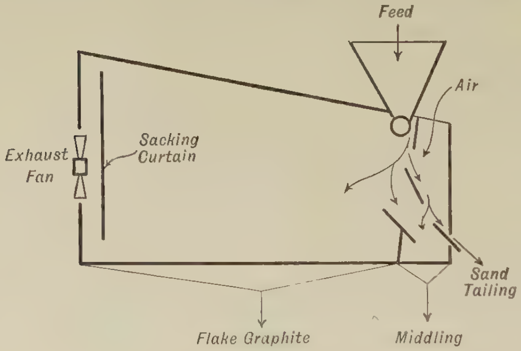


FIG. 396.

Dry-blower.—Diagram of the Exhausting Type. The design illustrated is largely used in finishing graphite-concentrate in Alabama and Ontario. Such air-classifiers are about 12 feet long, 6 feet high, and 3 feet wide (p. 577).

Aspirators.—Where it is required to separate dust, either to get rid of it, or being valuable to collect it, an exhausting fan is used to raise and carry it away. Such fans are often used to draw the hot gases through revolving driers, the current thus set up taking the dust with it; or to suck away the dust arising in process of grinding; or they may work in connection with special 'de-dusters' (Fig. 397). Exceptionally, in one type of dry concentrator, consisting of an inclined and slowly-rotating cylinder, so strong a suction is designedly created that the lighter gangue is removed in an upward direction, while the heavy mineral gradually descends to be delivered at the bottom.

Such aspiration is employed to remove the dust to which, for instance, the mineral carnotite is reduced when the sandstone containing it is crushed, this dust being recovered in 'cyclone' dust-collectors, the finest in bag-houses (Fig. 398). It was also the basis of the Goltra process tried on a commercial scale at Waukon, Idaho, in the beneficiation of an ironstone deposit consisting of nodules of hæmatite embedded in clay. Fed into the upper end of a large slightly-inclined revolving cylinder, the clay, becoming dried by the hot blast from a flame burning at the lower end, and falling then to powder, was carried away by that blast, the hæmatite alone travelling downward.

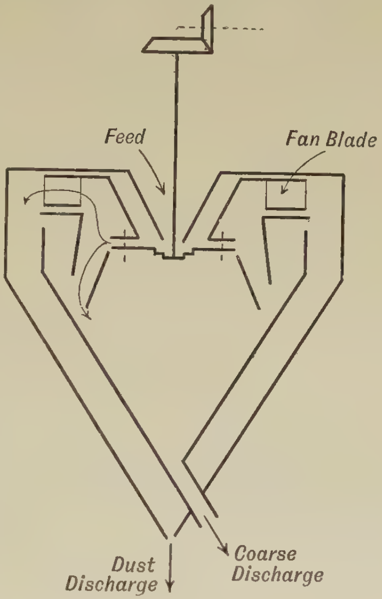


Fig. 397.

Dust Separator or De-Duster.—Diagram. The dry material is fed centrally into the cylindrical portion of a steel shell with steep conical bottom. There it is distributed centrifugally by a rapidly-rotating disc into a zone commanded by an air-current created by fan-blades moving with the disc. Submitted to this air-current the coarser particles fall directly into an interior shell, while the fine is carried upwards to fall between the two shells. This dust separator, which in fact is an air-sizing appliance, is largely used in the preparation of cement (p. 579).

APPLICATION OF PNEUMATIC CONCENTRATION

Pneumatic concentration to-day is used chiefly in the recovery of gold and diamonds from desert alluvial-deposits; in the refining of graphite concentrate obtained by flotation or otherwise; in the recovery of asbestos fibre which in process of crushing becomes fluffy while the enclosing rock goes to powder; and in the beneficiation of rare-metal ores—carnotite, for instance, and, formerly, molybdenite. Only to an unimportant extent has it ever been employed in the treatment of base-metal ores.

Dry-washing for gold has been largely, and still to some extent is, practised in Western Australia; in Sonora, Mexico; in Yuma County, California; in Oregon; in Arizona, and elsewhere. Generally, the material treated is loose enough to be at once screened and dry-washed, but sometimes, being cemented, it requires a previous disintegration. Satisfactory disintegration of such material is accomplished in the hammer type of mill, the Quenner mill, for instance, which so breaks up the valuable cement that it passes out through the barred walls, while the worthless boulders are thrown out stripped and smooth at the discharge end, a rim holding back any gold nuggets. It may also be accomplished in drum mills like concrete mixers, the larger boulders of the cement then becoming the crushing media (Fig. 399). After such dry-crushing, the fine material is, as a rule, treated on a pneumatic table, the Stebbins table, for instance. Where no such disintegration is required, and the fine gravel, after screening, has been treated on an ordinary dry-washer, the concentrate recovered

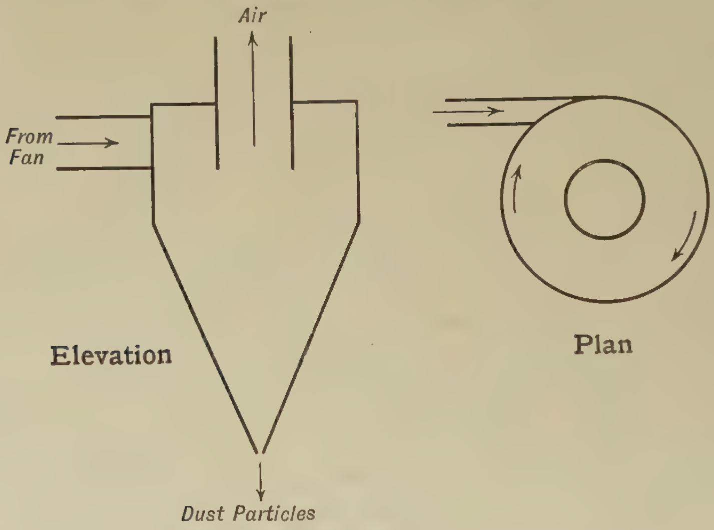


FIG. 398.

Cyclone Dust Collector.—Diagram. This apparatus consists of a short cylindrical shell with a steeply conical bottom; it is 3—9 feet diameter and 6—20 feet in height. Into the upper cylindrical portion the dust-laden air or gases are blown tangentially, developing a whirling motion; in their course they encounter deflecting plates, the whole effect being that the dust particles are thrown outward and downward, while the de-dusted air escapes upward at the centre (p. 579).

is usually cleaned from associated heavy sands, magnetite, ilmenite, garnet, etc., by dry-blowing in pans.

In the beneficiation of graphite ores, common practice in Alabama is as follows:[1] the ore containing, say, 2·25 per cent of carbon is broken and, after the necessary drying, crushed by rolls to about 16 mesh. This dry-crushed material is then screened or air-classified into sand containing 1·33 per cent, which goes to the waste dump; middling containing 3·5 per cent, which, after grinding, goes to the main flotation cells; and dust, which goes to separate flotation cells. The concentrate resulting from flotation, containing 40—60 per cent of carbon, is then sent to the finishing plant. In that plant the dried concentrate is usually fed to a dry-blower, which separates the flaky graphite by blowing it away from the associated impurities. This flake graphite in turn is carefully screened to separate the higher grades from the lower grades, and both from any remaining sand.

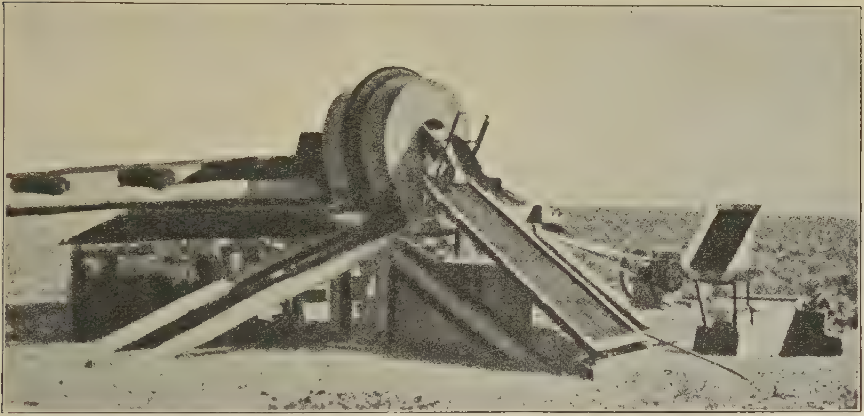


FIG. 399.

Dry-washing Plant at Baker, Oregon.—General View. The illustration shows drum disintegrator, grizzly, and gas-engine; in addition a half-inch screen and a Stebbins table were included. From a cubic yard of gravel about 15 lb. of concentrate was obtained (p. 580) (*M. & S.P.*).

The gradual development in the treatment of graphite ores is well illustrated by the experience in Quebec, Canada.[2] In the early days, before 1900, the milling scheme included breaking, stamping, and buddling, the resulting concentrate being dried, ground between buhr-stones, and then screened to remove sand and divide into grades. In that way a coarse flake containing 92—95 per cent of carbon was obtained, a fine flake of 90—92 per cent, and dust of 60—72 per cent, but the recovery was only 30 per cent of the graphite originally in the ore. In 1901 the Brumell 'skin-flotation' classifier was introduced, which gave a 55-per-cent concentrate to be cleaned and graded, and a recovery of 50 per cent. Then the Hooper table was applied to remove the coarse flake, the tailing being submitted to skin flotation; by this practice the milling concentrate contained 60 per cent of carbon, and the recovery was 70 per cent. In 1908 the Sutton-Steele table increased the grade of the concentrate to 64 per cent and the recovery to 85 per cent. Finally, in 1918 froth flotation in pneumatic cells increased the recovery to about 90 per cent, in grades varying from 40 per cent to 90 per cent of carbon. The milling plant then included breakers, rolls, conical mills, pneumatic flotation cells; and the finishing plant, shaking tables, drier, polishing rolls, mill-stones, and screens.

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[1] Dub, Pamphlet No. 3, War-Minerals Investigation Series, *Abst. M. and S.P.*, Mar. 1, 1919, p. 283.

[2] Brumell, *E. & M.J.*, Feb. 28, 1920, p. 548.

The pneumatic recovery of carnotite from the sandstone containing it, is made possible by its pronounced friability. This radium-uranium mineral occurs in South-west Colorado and South-east Utah, as the binder between sand grains.[1] Crushed to the size of these grains, most of the carnotite goes to dust while the sand yet remains granular. After this crushing the fine material, which may be recovered wet in the form of settled slime or dry in the form of collected dust, is the concentrate.

Recovered dry, the complete treatment is as follows: the ore is hand-broken and sorted, then machine-broken to $\frac{1}{4}$ in., dried in rotary drier, the dust from breaker and drier being sucked away by fan and blown into a cyclone dust-collector, the finest dust passing thence through a bag-house before escaping to the atmosphere. From the drier the hot ore is screened over 10 mesh, the oversize going to rolls and then over the screen again, the undersize going directly to Raymond mills. In these mills, which grind to 80 mesh, the carnotite is released, a fan drawing it away, while the granular product remains impoverished. The dust so drawn away is settled in a second cyclone collector, whence the finest dust passes to a tubular dust-collector (Fig. 400).

Treating ore containing 0·85 per cent of U_3O_8, the average value of the four separate dust collections was about 3 per cent; the sandy tailing, on the other hand, contained but 0·37 per cent. The dust drawn away from the grinding mill and collected in the second cyclone, constituted the great bulk of the valuable product.

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[1] Kithil and Jones, Bull. 103, Mineral Technology 11, U.S. Bureau of Mines, 1917; *Abst. M. & S.P.*, July 13, 1918, p. 55.

In 1917 at the tungsten mine, Hill City, South Dakota, the ore crushed by rolls to a proper size went to a pneumatic jig for the removal of mica, after which it was concentrated by water.

The concentration of molybdenite in Ontario was, till the advent of flotation, accomplished dry in a scheme which included sorting, breaking, rolling to 20 mesh, drying, screening over 60 mesh, and treatment on pneumatic tables.

Other interesting applications of pneumatic separation occur in association with dry magnetic-separation. At Mineville, New York, tailing from magnetic concentration, consisting mostly of hornblende and quartz, is treated on Hooper and on Sutton-Steele tables to recover the apatite; mixed with barium sulphate this apatite forms a first-class fertilizer marketed as 'barium phosphate.' At the Ringwood iron mine, New Jersey, dry-jigging has been employed to recover martite, the haematite pseudomorph of magnetite, which, being only feebly magnetic, escaped with the non-magnetics.

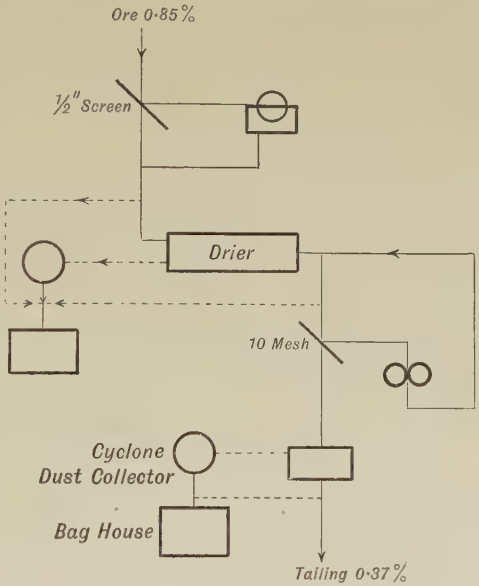


Fig. 400.

Pneumatic Concentration of Carnotite.—Flow-sheet. The ore which assays 0·85 per cent of U_3O_8 is crushed to 10 mesh, when the dust is withdrawn. The dust amounts to about 20 per cent of the original ore, and it assays about 3 per cent of U_3O_8 (p. 582).

Centrifugal Separation

If dry crushed-material be fed on to a rapidly rotating horizontal disc, the particles are caught by friction, and then by centrifugal force thrown off radially. Were there no resistance to their flight, all the particles, whatever their size and density, would follow the same trajectory, but when flight is at a high velocity, friction with the air becomes a discriminating resistance, keeping back the fine and less-dense particle. Accordingly, if such a disc be surrounded by concentric troughs, the large dense particles will be found in the outermost trough, and the small less-dense particles in the innermost, the intermediate troughs being occupied by small dense particles and large less-dense particles; there would, in fact, be a classification similar to that resulting from fall in water.

Centrifugal separators embodying the above idea, such as the Clarkson-Stansfield, failed because the products were not sufficiently clean, and because the middling product bulked too largely (Fig. 401). This was so even when the air resistance was augmented by a centripetal current, which, drawn from the periphery, carried the lighter dust inwards, and down through a pipe at the centre, as in the Pape-Henneberg separator, an appliance some 20 ft. in diameter.

The similar free projection of water-borne material offers no possibilities even of similar separation, since the water would be scattered, the continuity of the medium would be broken, and confusion would result; accordingly, the only possible chance of applying centrifugal separation to such a pulp lies in confinement.

If an ore-pulp be confined in a tube so attached to a vertical spindle that its lower and closed end can swing out horizontally when that spindle is rotated rapidly, then upon such rotation the larger and denser particles will quickly fly to that end, leaving the smaller and less-dense particles in a more central position, while the open end of the tube will be occupied by clear water.

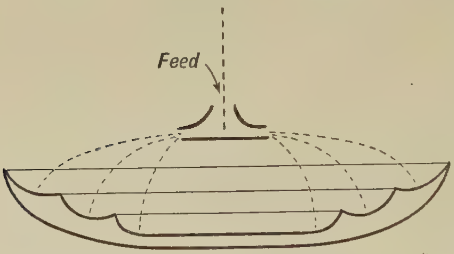


FIG. 401.

Clarkson-Stansfield Concentrator.—Diagram (p. 584).

The laboratory centrifuge operating on this idea, consists of two cups fastened to a horizontal spider fixed on a vertical spindle. Into these cups test-tubes are placed, the whole system being designed to be driven 1000—2000 r.p.m. If into these test-tubes a fine suspension of mixed mineral and gangue be placed, and the centrifuge be operated, it will be found, after a few moments, that the mineral has become tightly packed at the bottom of the tube, with the gangue above it, and clear water at the top. Provided all the material were fine, say, finer than 200 mesh, the line of demarcation between mineral and gangue would be fairly definite, and, by graduating the end of the tube, it would be possible to read off an approximate figure for the amount of mineral present; but if the range of size were greater, the coarser particles of gangue would be packed among the finer particles of mineral, and no definite separation would be made. In the absence of any mineral, an approximate figure could be obtained for the amount of granular solids present in the suspension.

The separation of mineral and gangue accomplished in a centrifuge, nevertheless, is better than that generally effected by a classifier, or by ordinary fall in water. While the force making for settlement is a large multiple of that available in gravity settlement, the resistance to settlement remains the same, and, accordingly, not only is settlement completed in a fraction of the time, but flat mineral-particles such as would probably be lifted to the overflow of a classifier, would in a centrifugal machine be forced to settle ; moreover, in such a machine the eddies and backwaters which militate against the close work of classifiers would no longer be capable of disturbing an ordered settlement.

The accelerating forces acting on a particle of diameter D and density δ suspended in water, are, under gravity and centrifugal force, respectively,

$$\frac{\pi}{6}D^3(\delta-1)\omega, \text{ and } \frac{\pi}{6}D^3(\delta-1)\omega\frac{V^2}{gr},$$

where V is the linear velocity of the particle around the circle of radius r, and ω is the specific weight of water. Assuming r to be 1 ft. the centrifugal factor V^2/gr would be about 100, at which rate the terminal or settling velocity under centrifugal force would be one hundred times that under gravity. It will be noted, however, that the relative rates of settlement of different minerals and gangue are not altered.

With the above points in its favour, and with centrifugal force so readily invoked, it might appear that centrifugal separation had considerable possibilities, particularly in the treatment of slime. The truth, however, is, that in the absence even of moderate success, centrifugal separation of mineral from gangue was discredited even before the victorious entry of flotation. The difficulties encountered were to some extent in the mechanism necessary for so high a rotative speed, but they lay also in the fundamental disability of the short available flight, from which it resulted that the solid particles packed themselves dense and immovable before the mineral had become disentangled from the gangue. There is also the difficulty of arranging a continuous discharge without breaking the continuity of the water. Accordingly, the only field in which centrifugal separation appears yet to retain some possibilities, lies in the separation of granular suspensions from colloidal material, and from water. Among others the following separators have been tried.

Peck Centrifugal Separator.—This machine consists of two pans with slightly sloping sides, set one within the other, upon the same central spindle, around which they are independently driven in the same direction (Fig. 402). Fed into the inner and smaller pan, the aqueous pulp passes through the bottom into the space between the two pans, the mineral then bedding itself upon the inner wall of the outer pan, while the gangue is carried by the water to overflow through appropriate nozzles around the lip; during the period of settlement the outer pan runs faster than the inner. When the bed of mineral is such that the machine may be considered to be loaded, the feed is stopped and clear water is introduced to wash the collected material, after which the speed of the outer pan is lowered to that of the inner, and the mineral is removed by strong water-jets. These cyclic alterations are effected automatically.

Peck's concentrator worked for some years, 1910—1912, on trial at Anaconda, Montana, making a commercial concentrate. The results, however, were not satisfactory, those from the ordinary round-table being better. It was also tried, but not adopted, at Cananea, Mexico.

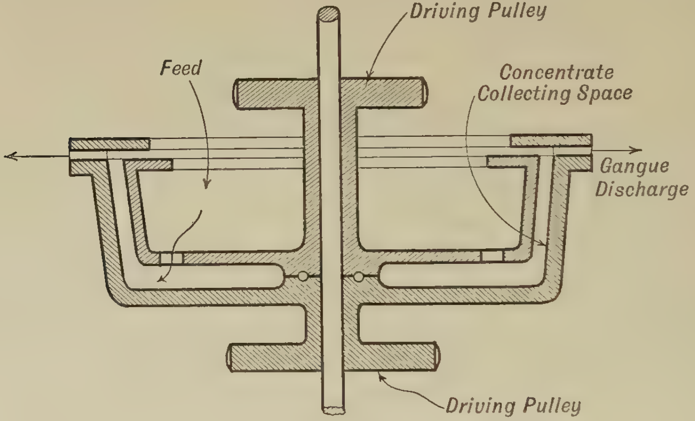
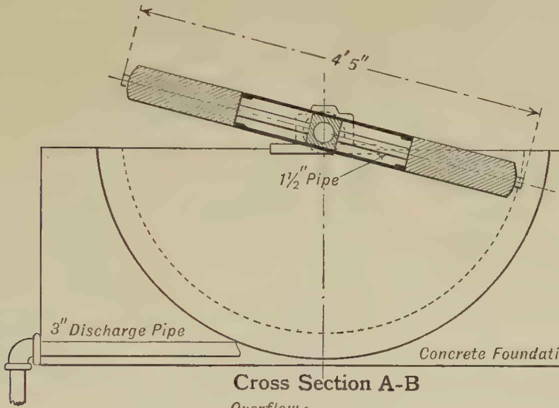


FIG. 402.

Peck Centrifugal Concentrator.—Diagram (p. 585).

Laist Centrifugal Separator.—This machine consists of radial chambers mounted on and revolved by a horizontal shaft (Fig. 403). Through one hollow end of this shaft the aqueous pulp to be separated is fed into the chambers, while through the other end the water flows away. Each chamber, at its pointed outer extremity, has a narrow aperture or spigot through which the settled material is continuously discharged. Within the chamber itself baffles check the flow, and direct the solids to the point of discharge. The speed is 600—1200 r.p.m., and the outside radius of the chamber about 2 ft. Fed with pulp containing about 5 per cent of solids, the consistency of the discharge varies with the diameter of the discharge aperture, and with the rate of feed; with an aperture of $\frac{1}{6}$ in. the discharge contains 30 per cent of solids, a percentage which quickly diminishes as the aperture becomes larger with wear. To resist wear these apertures are best made in plugs of sintered alumina. This machine was tried at Anaconda in 1913—1914 as a separator of granular slime from colloidal material and excess water, but was not adopted. It is a continuous machine.



Cross Section A-B

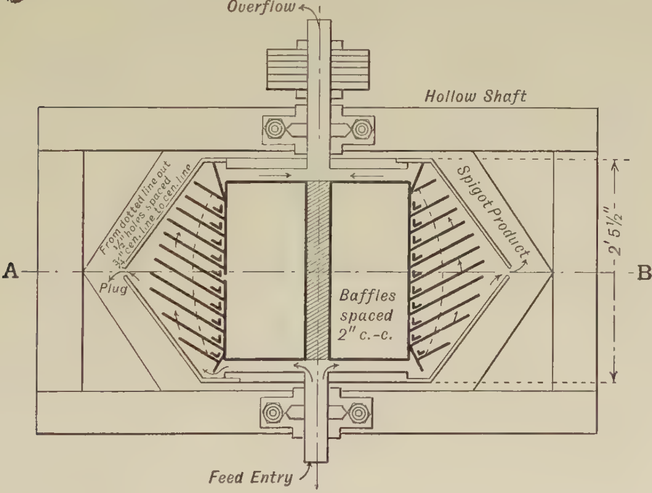


FIG. 403.

Laist Centrifugal Separator.—Plan and Cross-section. Two chambers in the same diametral plane, one on either side of the centre, are shown; as many as six such pairs could be accommodated (p. 586).

Mauss Centrifugal Filter.—This separator of granular from colloidal material, consists of two bottomless buckets held diametrically to a central spindle within an enveloping pan, in such a manner, that while the pan and buckets make 500 revolutions per minute around the spindle, the buckets also make a slower movement around their own axes; into these buckets the suspension is fed (Fig. 404). Bedded by the first movement upon what at that moment was the farthest wall of the bucket, the packed solids are by the second movement gradually brought round to an inner position, where, lacking support, they are thrown off and discharged; the liquid in the meantime continuously climbs the bucket wall and overflows peripherally. This particular continuous machine was used at the

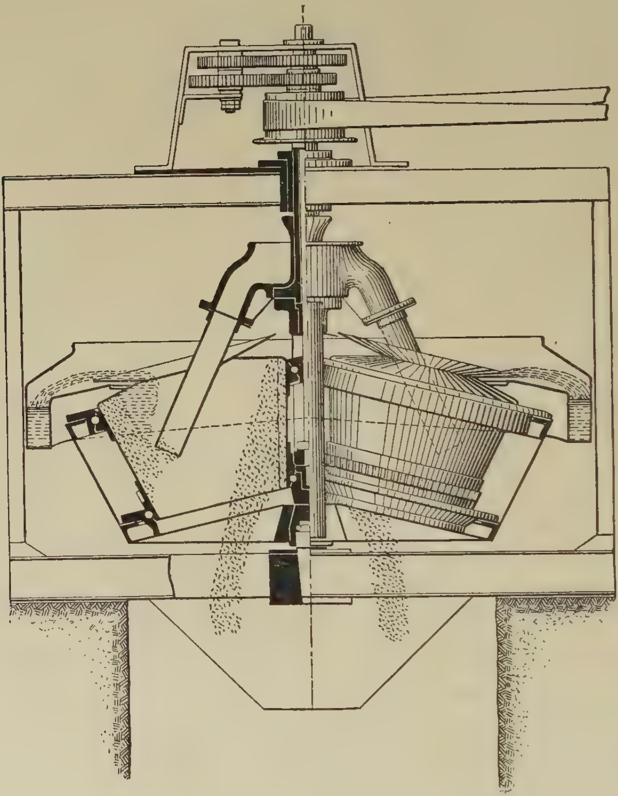


Fig. 404.

Mauss Centrifugal Filter.—Half-sectional Elevation. The pan is about 4 feet diameter and makes about 500 revolutions per minute. The buckets make 100 revolutions per minute (p. 588) (*E. & M.J.*, September 20, 1913; May 17, 1919).

Zaaiplaats tin mine, Transvaal, to separate granular material for treatment on slime tables.

Gee Separator.—This appliance consists of a vertical cylinder about 3 ft. 6 in. diameter and 4 ft. long, mounted on a vertical spindle driven at about 700 revolutions per minute (Fig. 405). Through a central opening in the top cover of this cylinder the pulp is fed; on the walls the solids build up, while the clear water is discharged centrally through the bottom cover. A depth of about 4 in. on the walls thus becomes the collecting space; but as the coarser and greater bulk of the material settles immediately, the upper portion of the collecting space becomes filled while the lower portions yet have but a light covering (Fig. 405). When so filled the machine is stopped and the sediment removed. The operation accordingly is intermittent and not continuous; under good conditions about four charges each of about half a ton can be collected per hour. To facilitate discharge the collecting space is divided by radial vanes held in place by rods, while removable liners hug the walls. Upon these latter the material packs, the vanes dividing the circle into convenient sections. This separator has been tried in Cornwall for the separation of china clay from the watery suspension in which it arrives from the pit.

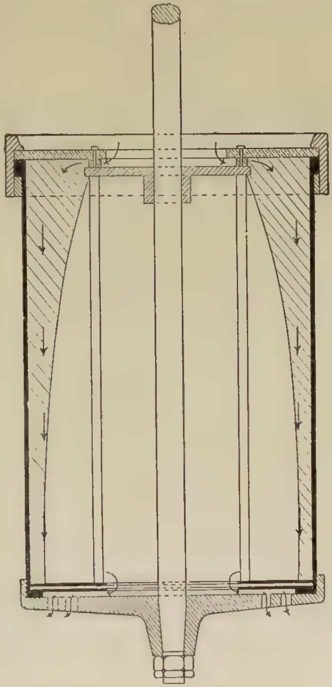


FIG. 405.

Gee Separator.—Diagrammatic Section. The hatching indicates deposited material. The thick end of this deposit is of relatively granular material; the thin end of the finest clay (p. 589).

Trent Centrifugal Separator.—This separator embodies some original ideas. A circular centrifuge on a hollow vertical shaft, has a narrow discharge aperture right round the periphery. This centrifuge revolves horizontally in a chamber filled with water under a pressure sufficient to force entry into the centrifuge at a rate under control by a valve on the water service. This rate is arranged to permit only the desired particles to pass out, while the pulp stream, drawn up the hollow shaft on which the centrifuge is fixed, carries the finer particles forward and away (Fig. 406).

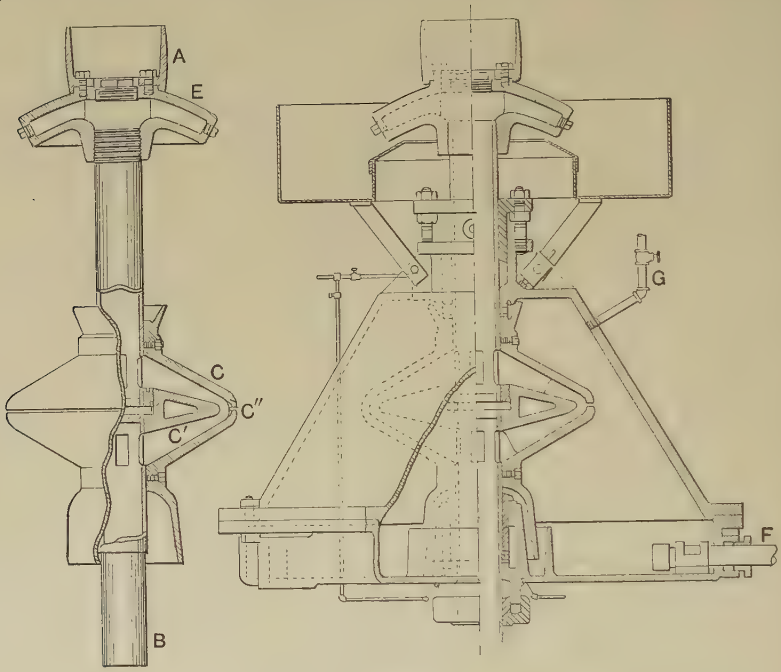


FIG. 406.

Trent Centrifugal Separator.—Part-sectional Elevations. A is the driving pulley, E is the centrifugal pump which sucks the feed up the hollow vertical shaft, C is the centrifuge, C′ a diaphragm within the centrifuge compelling the entire rising-stream to present itself for selection at the aperture C″. Through this aperture and against water entering from outside, the mineral particles force their way into the pressure chamber outside, out from which they eventually pass through the spigot F. The use of water inflowing at C″ is only made when mineral particles are to be separated from gangue; otherwise, when it is required only to separate solids from water, no water passes into the centrifuge, but some passes out with the solids, the amount so passing out being determined by the aperture at F (p. 589). (*E. & M.J.*, June 20, 1912, p. 1259.)

CHAPTER XIII

HEAT TREATMENTS IN ORE-DRESSING

Though heat treatments of ores, smelting for instance, generally involve chemical change and primarily are metallurgical operations, certain specific heat treatments may be regarded as coming within the purview of dressing, in that they are either preparatory to other purely-dressing operations, or are preparatory to despatch to the metallurgical works. Such treatments are:

Drying and Dehydrating, to remove water.
Calcining, to remove carbonic acid, etc.
Calcining, to release and disintegrate.
Magnetic-roasting, to make magnetic.
Fractional-roasting, to deaden the surface.

Drying and Dehydrating

Drying of the crude ore is undertaken: to remove excessive moisture from iron ores preparatory to shipment, whereby freight is reduced and the ore rendered more acceptable to the smelter; to prepare crude ore, chiefly magnetite ore, for dry magnetic separation; to prepare ores, chiefly refractory gold ores, for dry-crushing, previous to roasting and cyanidation; and exceptionally to prepare ores for pneumatic separation. Drying of flotation concentrate is undertaken to remove excess water, and drying of water concentrate in preparation for magnetic or electrostatic separation.

Drying of Iron Ores for Shipment.—Much of the ore of the Mesabi and Marquette Ranges, Minnesota, contains 15—20 per cent of water. To send such ore to the Lake Erie ports for smelting means the payment of freight upon the contained water, and a poor market upon arrival. Such ore, accordingly, is better dried.

Drying takes place in large cylindrical kilns, slowly revolving round an axis slightly inclined to the horizontal (Figs. 407, 410, 413). These kilns are perhaps 60 ft. in length and 7—8 ft. in diameter; they are supported on rollers, and driven at the rate of about one revolution per minute by the engagement of a pinion with a toothed wheel enveloping the kiln at a medial position. In revolution, the coarsely-broken ore fed at the upper end is raised by longitudinal lifters, till as these overturn it falls, arriving at the bottom a little forward of the position at which it was picked up, the inclination of the drier towards the discharge being about three-quarters of an inch per foot (Fig. 408). Fall takes place in the hot gases coming from a stationary firebox at the same upper-end ; the moist ore is thus exposed to the greatest heat. The fuel used is either ordinary coal, powdered coal, or fuel-oil,

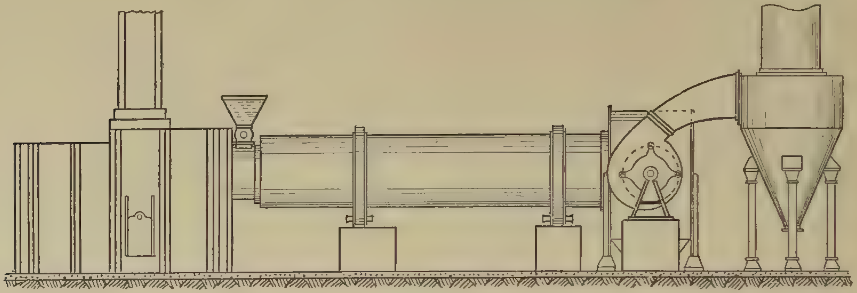


Fig. 407.

Rotary Drier for Crude Ore (Buttner).—General View. The ore, fed through the hopper at the firebox end, travels with the flame to the discharge end, where it falls and is removed by a screw-conveyor. The draught created by a fan at the discharge end blows the gases into a 'cyclone' dust collector, on their way to the stack (p. 591).

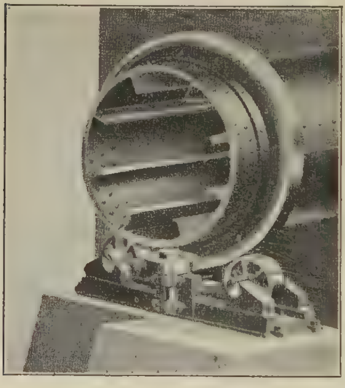


Fig. 408.

Rotary Drier.—Cross - section, showing longitudinal lifters (p. 592).

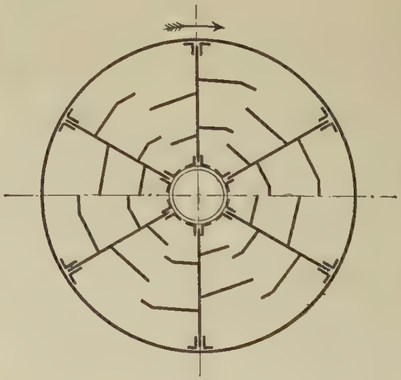


Fig. 409.

Rotary Drier.—Cross-section, showing longitudinal diaphragms which increase the cascading of the ore (p. 593).

the normal temperature of combustion, which would be too high for the purpose, being reduced by air dilution to about 800°—900° C. Passing down the kiln the ore becomes drier and the temperature of the gases lower, the temperature at discharge being generally less than 100° C. The necessary draught is maintained by a suction fan, which, if necessary, may blow the gases into a dust collector.

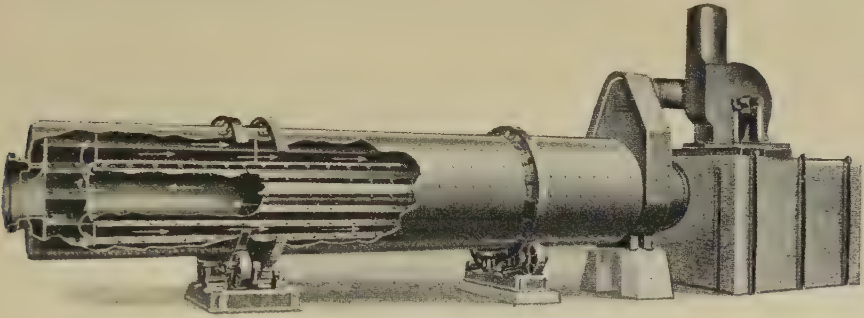


FIG. 410.

Rotary Drier for Crude Ore (Ruggles-Coles).—Part-sectional Elevation. A double-shell kiln is illustrated. At the far and upper end is the firebox; on top of the firebox is the fan which sucks the gases from the annular space between the two concentric shells, and delivers them to the stack. Into the same annular space the crude ore is fed at the upper end, the feed chute not being visible in the illustration. It travels down against the up-coming current till at the lower end it is raised by scoops to be discharged centrally. The up-coming current is the return of the hot gases after their downward flow through the inner shell (pp. 591, 593).

Instead of this simple arrangement the interior may be divided by longitudinal diaphragms, from which and on to which, in the course of revolution, the ore is continually cascading (Fig. 409). Or, the drier may consist of an inside and outside cylinder with lifters externally on the former and internally on the latter (Figs. 410, 411). With this last arrangement the hot gases pass down the inside cylinder, to return between the two cylinders, coming there into intimate contact with the ore passing down the same annular space. The discharge end then is closed to the gases by an end plate associated with a scoop discharge for the ore. In addition to securing intimate contact of hot air and ore, this arrangement has the

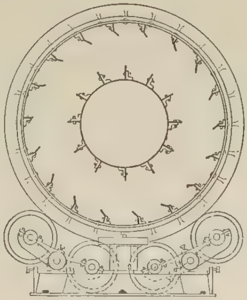


FIG. 411.

Rotary Drier. — Cross Section of Double-shell Drier (p. 593).

advantage that the exhaust gases pass away cooled after immediate contact with moist and cool ore.

The passage through such driers generally takes about 45 minutes, by which time the moisture has been reduced to about 7½ per cent, less than which would render the ore friable and create too much dust. The capacity of such a drier is about 300—500 tons per day, and the total cost of drying about 12—15d. per ton. Of this cost that of the fuel is about one-third, the efficiency in the use of the fuel varying between 50 and 75 per cent.

Drying on the Mesabi is only undertaken during the shipping season, when the lakes are ice-free. The dried ore not only has a more certain market and never freezes during transport, but in addition it commands about 20—24d. more per ton, so that there is a decided profit by drying. The cost of the necessary plant when on a scale of about 1000 tons per day is at the rate of about £10 per ton dried per day, the above figure for total cost including interest and amortization on such a figure.

Similar drying is undertaken on Cuban iron ores for shipment to the United States. These ores are not only dried, but to a large extent nodulized. The driers are 130 ft. long and 10 ft. diameter; taking two minutes to make a revolution, discharge is delayed, so that their capacity is only about 200 tons per day.

Drying of Magnetite Ores for Magnetic Separation.—Magnetite ores do not as a rule contain much water, nor does ordinary dampness interfere with the magnetic separation of coarse magnetite. The drying of magnetite ores is therefore limited to the removal of, say, 5 per cent of water from the finer portion of the broken ore, that is, material finer than 1—3 in. Being a limited operation, the drier usually employed is a simple brick tower about 5 ft. square in section and 45 ft. in height. A number of inclined baffles fixed in this tower delay the fall of the ore fed at the top; these baffles are of cast-iron bars laid close together, the directions of inclination of any two successive baffles being at right angles. At the bottom of such a tower the hot gases from a firebox enter and rise; having fallen through these gases the dried material discharges through a gate. At the top is the feed inlet, kept filled with ore, and the suction of an exhaust fan which carries away the gases and the dust.

Drying and Dehydration of Gold Ores.—The drying of gold ores in preparation for dry-crushing and eventual cyanidation is not uncommon. At Kalgoorlie, Western Australia, and at the Cam-and-Motor, Rhodesia, it was necessary, in order to reduce the moisture in the crude ore to about 1 per cent, or otherwise ball-mill crushing would have suffered by the screens becoming clogged. At the Connemara mine, Rhodesia, the fine portion of the ore, containing a good deal of hydrated iron oxide and other hydrates, was dried in a rotary drier to the advantage of the subsequent recovery by cyanidation. Tests made at the Buckhorn mine, Nevada, where the ore was very clayey, showed that by drying the ore at a temperature sufficient to drive off the combined moisture, better recoveries were possible afterwards; such drying appears so to alter the physical condition of any clay present, that the gelatinous condition does not arise to the detriment of subsequent percolation, dissolution, and washing. In a drier, combined water appears to be removed just as readily as ordinary moisture.

Drying as a preparation for the pneumatic separation of rare ores was instanced when describing the recovery of carnotite by that means of separation.

Drying of Concentrate.—As already described under flotation, the mineral-laden froth from the flotation boxes is generally settled in mechanical settlers. From these, if the mineral be not too slimy, a product containing about 40 per cent of water is readily obtained. It is the practice now to filter this concentrate by continuous filters and thereby to reduce the moisture to about 12—15 per cent, in which condition it is delivered to the concentrate bins to drain while awaiting shipment. Where, however, the concentrate is slimy, the filtered product will still contain as much as or more than 25—35 per cent of water, and be a sticky material difficult to handle; such slimy concentrate is best dried. The Lowden drier, already described (p. 464), readily brings the moisture down to about 12—15 per cent; or, with less slimy flotation concentrate filtered to 15 per cent moisture, readily to 5 per cent, though in transport there would then be risk of loss by dusting. Ordinarily, just over a hundredweight of coal per ton of dry concentrate is required.

On the other hand, a satisfactorily low moisture-content has sometimes been obtained by directly drying the thickened underflow from the mechanical settlers, either in shallow tanks with steam coils in the bottom, or upon floors. Suitable tanks are about 15 ft. square and 3 ft. deep; into such the thickened underflow is run and allowed to settle; the supernatant liquor is decanted, and steam turned on; after 24—36 hours' steaming, the moisture will be reduced to 8—10 per cent, and the bed of dry concentrate, about a foot thick, will be in a condition to be dug out. Filtration is, however, cheaper.

At the Tul Mi Chung mill, Korea,[1] the flotation concentrate, leaving the thickener with about 33 per cent moisture, is dried on floors each of which is 40 ft. long and 15 ft. wide, and has staggered beneath it flues having a total length of 300 ft. from firebox to stack. The floor itself is made of concrete slabs, 2½ in. thick, reinforced with barbed wire, the joints being of the ship-lap type to prevent concentrate falling between; it slopes about ½ in. per foot towards the stack where the ore is fed, this slope giving opportunity for water to drain off. As the concentrate dries it is raked towards the firebox, the moisture in the finished product being well below 1 per cent; drying is continued to such completeness, and such completeness is possible without subsequent loss by dusting, because the concentrate is bagged for shipment. Each of these floors dries 5—6 tons per day of flotation concentrate, 97 per cent of which is finer than 200 mesh, at a cost of 3s. 6d. per ton for drying and bagging.

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[1] Weigall and Mitchell-Roberts, *M. & S.P.*, Dec. 6, 1919, p. 815.

Water concentrate must be dried if it is to be submitted to electrostatic separation, or unroasted, to dry magnetic separation; drying is also often undertaken before roasting. Such drying may be done on floors, plates, or on the top of the roaster, but the product is more uniform if the operation is conducted in a rotary drier. A drier of relatively small dimension, say 3 ft. diameter and 30 ft. long, suffices; in such, it is the common practice for the material to progress towards the flame, the moist concentrate being fed at the stack end, and the finished product being discharged at the firebox end (Fig. 412).

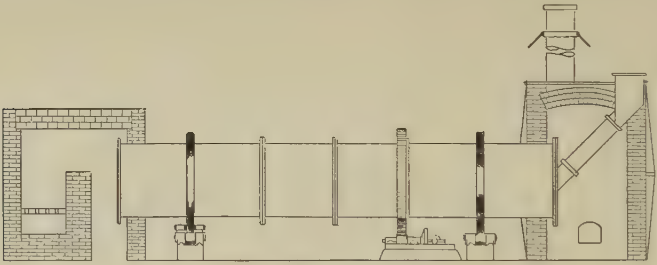

Fig. 412.

Rotary Drier for Concentrate.—Diagrammatic Elevation. Feed is at the stack end, discharge at the firebox end; the material travels towards the flame (pp. 596, 601).

Exceptionally, at the Braden copper mine, Chili, the flotation concentrate is not only dried, but nodulized in rotary kilns heated by oil burners to 950° C. at the feed end. At that temperature the granular concentrate becomes sticky and the rolling motion causes balling; the sulphur at the same time becomes reduced from 28 per cent to about 18 per cent.

Finally, driers, like pneumatic concentrators, are largely employed in the preparation of non-metalliferous minerals, asbestos, graphite, mica, bauxite, etc.

CALCINING TO REMOVE CARBONIC ACID

Calcining, popularly, is to reduce to quicklime or to render a substance friable, by roasting or burning; hence, though calcining and roasting are used indifferently, in a stricter sense the former is characterized by the disengagement of carbonic acid, and has a physical change in view. As such it is employed to reduce the weight of siderite in preparation for shipment, giving at the same time a material of higher iron-content and better physical condition for the blast furnace. This calcined material, however, gradually absorbs moisture, so that if the delay be long, the moisture-content may again be as much as 8—10 per cent.

In a pure state siderite contains 48·3 per cent of iron and 37·9 per cent of carbonic acid; it is, however, generally mixed with the carbonates of calcium and magnesium, some alumina and silica. A fair average-figure for the iron-content of carbonate ore from the Inferior Oolite in Yorkshire, after heating to 100° C. to expel moisture, is about 33 per cent. By calcination this figure is increased to just over 45 per cent; the change is from ferrous carbonate, $FeCO_3$, to ferric oxide, Fe_2O_3, thus: $2FeCO_3 + O = Fe_2O_3 + 2CO_2$. Drying taking place concomitantly with calcination, the weight of the raw ore is reduced 25—30 per cent.

Rhodocroisite, the carbonate of manganese, when existing in such amount as to warrant exploitation, is also submitted to calcination.[1]

The oxidized ores of zinc, and particularly the carbonate, may likewise be calcined with advantage in lower weight and higher zinc-content, the combined water being disengaged as well as carbonic acid. At the same time any siderite present is rendered magnetic, and removable subsequently by magnetic separation.

Finally, when magnesite, the carbonate of magnesium, is calcined it loses about half its weight, becoming then the material for the basic linings of steel furnaces. Being at the same time and by reason of contained iron, rendered magnetic, it can be magnetically separated from any impurities should such separation be desired.

Calcination to remove carbonic acid is conducted in fixed vertical kilns, or in rotating horizontal kilns similar to rotary driers. In the former, progression of the material fed at the top is by direct fall to the bottom, the rate of progression being determined by the rate of withdrawal below; in the latter, progression is down the gentle inclination, this progression being promoted by the successive rise and fall of the material, as the cylinder revolves. With the vertical kiln the heat may be introduced as hot gases from fireboxes at the side, or arise directly within the kiln by the combustion of fuel interbedded with the ore, about 8 per cent of fuel being required in either case; these kilns are about 45 ft. high and 10 ft. outside diameter, the firebrick lining being about 15 in. thick. With the rotary kilns, as already described, the heat enters with the gases of combustion from a fixed firebox or a fixed burner; the advantage of this type is that fine material cannot obstruct the free passage of the hot gases.

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[1] Moreing, *Trans. I.M.M.*, Vol. II., 1894, p. 257.

In Great Britain and in Europe generally, vertical kilns are used; in America the rotary kiln (Fig. 413). At the Magpie siderite mine, Ontario,[1] the calcining kilns are 125 ft. long by 8 ft. diameter, lined with 9 in. of firebricks; powdered slack is the fuel, and the temperature obtaining within the first 20 ft. of the kiln is about 1100° C. The ore to be calcined is broken to about 3 in.; in the operation it loses about 30 per cent of its weight and something of its size; the colour also changes from brown to nearly black; coolers are used after calcination.

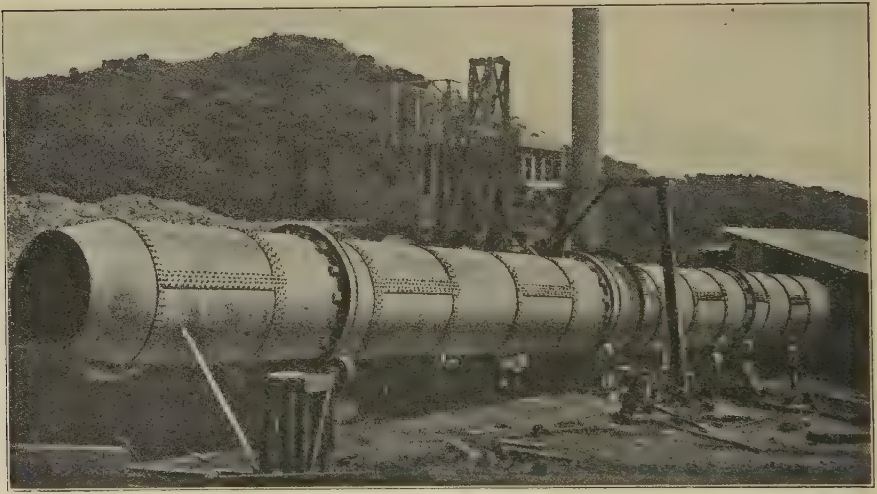

FIG. 413.

Rotary Kiln for Magnesite.—In process of erection. This kiln, as will be realized readily, is about 130 feet long and 10 feet in diameter. The near end will be the feed end and the cool end (pp. 591, 598).

Similar, though generally smaller, rotary kilns are used with magnesite. Broken to 1 in., this material is fed into a kiln about 60 ft. long by 6 ft. in diameter, making about one revolution per minute; therein it remains for about 45 minutes. Travelling slowly to meet the flame, it is discharged at a cherry-red heat necessitating special cooling. The capacity of such a kiln is about 60 tons per day.

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[1] Hasselbring, *Trans. Can. M.I.*, Vol. XX., 1917, p. 325.

CALCINING TO DISINTEGRATE AND RELEASE

Ages old is the practice in Cornwall to burn the tin concentrate before submitting it to the more careful enriching-operations conducted in the tin-yard. This burning breaks up pyrite, mispickel, etc., that these impurities may, in a subsequent water-treatment, be removed in suspension or streamed away. At the same time any cassiterite or wolframite held by such sulphides is released, becoming to a large extent recoverable by water; any cassiterite and wolframite held together by sulphides or other cement fall apart, to the more complete magnetic-separation of these two minerals afterwards; and the arsenic, disengaged in the form of arsenious oxide, As_2O_3, becomes recoverable in appropriate flues, while the sulphur fumes escape up the stack. This operation is described as calcining, and the appropriate furnaces as calciners. The concentrate submitted to the operation contains variously from 7·5 per cent to 30 per cent of tin, the lower percentage connoting much arsenic or wolframite to recover and/or much sulphur to remove.

Similar calcination is conducted in Bolivia, in the Malay Peninsula, and elsewhere, wherever there are impurities to remove from cassiterite, or wolframite and cassiterite to separate from one another.

The temperature in the calciner should not exceed 800° C., particularly if wolframite be present, because above that temperature the larger grains burst and excessively fine mineral is produced; incipient fusion will also develop. The best temperature is 600°—700° C.[1] Where considerable pyrite or mispickel is present, fritting of the mass begins even at a lower temperature, so that it then becomes necessary to conduct the calcination in two stages, an earlier stage at a lower temperature, and a final stage at a higher temperature, the material being dressed between the two stages. Deposition of arsenious oxide begins when the temperature of the gases in the flues has fallen to about 170° C., and is practically complete at about 140° C.

The calciner favoured in Cornwall is one with a revolving hearth, the Brunton calciner (Fig. 414). This hearth, about 12 ft. diameter and covered with firebrick, is supported upon an iron framework radiating from a central driving-spindle; in shape it is a flat cone, with a slope of about 1 in 10. Revolution, at the rate of about three times per hour, takes place in a firebrick chamber, the arch of which is about 18 in. above the hearth. In the centre of the arch is a feed-cone which connects with a drying floor above. This cone is kept filled with material to prevent the escape of fume; up through it, the driving spindle extends to a bearing above; within it, radial stirrers moving with the spindle ensure a regular feed.

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[1] Taylor, *M. Mag.*, May 1918.

Two fireboxes, spaced with a 60° arc between them on the periphery, connect directly with the combustion space above the hearth; at a convenient point away from these boxes is the exhaust flue leading to the depositing chambers, and eventually, after a traverse of 700—1000 ft., to the stack. Fixed in the arch are three radial rows of ploughs, which, as the hearth revolves, gradually plough the material from the centre to the periphery, this passage taking 6—12 hours. At a point on this periphery opposite the fireboxes is a controlled opening for discharge, the calcined material being ploughed off by the outside plough of the row of ploughs radiating to this point.

Such a calciner roasts about 4 tons of coarse concentrate per day, but less of fine, and consumes about 200 lb. of coal per ton of material roasted. In the first part of the depositing flues some fine cassiterite settles; the arsenic settles later, black with unconsumed carbon and containing about 75 per cent of arsenious oxide. A subsequent refinement eliminates the carbon and raises the arsenious oxide to 98 per cent, in which condition the product is marketable. Where the amount of arsenic is considerable the flues are cleaned about once a month. The fine cassiterite recovered is generally dressed separately from the milling concentrate. Though there is but little draught, some of the finest cassiterite undoubtedly is lost with the escaping gases.

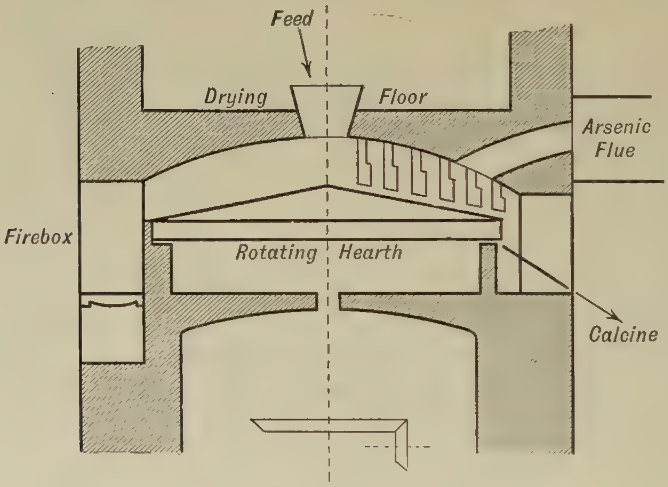

FIG. 414.

Brunton Calciner.—Diagram (p. 599).

The Brunton calciner is favoured in Cornwall probably because of its gentle operation and its suitably low capacity; several are usually required, the coarse concentrate, the fine, and the material to be re-roasted, all being treated separately. The resultant calcine is fairly sweet, but with sparse air and quiet rabbling some sulphur and arsenic remain. Such conditions, however, render the iron oxide sufficiently magnetic to be removed in a weak field, leaving any wolframite to be separated uncontaminated in a strong field. This particular furnace, modified to suit local circumstances, has also been used for the recovery of arsenic from lead fume at Midvale, Utah, where, in the presence of lead, the lower temperature of 475° C. was employed.[1]

The rotating-cylinder type of furnace, represented by the Oxland calciner, has also seen considerable use in Cornwall and Devon. This calciner, very similar in design to the small rotary drier, is a steel-plate cylinder about 30 ft. long and 3 ft. internal diameter, lined with firebricks, four longitudinal rows of which project to act as lifters (Fig. 412). This cylinder is encircled with tyres which find support on rollers laid to give a slight inclination; it is encircled in a medial position by a toothed wheel, through which rotation at the rate of about six revolutions per hour is effected. It communicates at its lower end with a firebox and at its upper end with flues and stack. The concentrate to be calcined is fed through a chute at the upper end, this chute being kept filled with green concentrate to prevent the escape of fume; the top of the fume-box around this chute makes a good drying-hearth. Travelling slowly down the cylinder with every revolution, the calcined material at last falls into a vaulted chamber fashioned in brickwork alongside the firebox. This calciner has a larger capacity than the Brunton, being capable of treating 12—20 tons per day; it also consumes less fuel per ton of material roasted, the amount being only about one hundredweight. The operation is, however, less gentle, dust being created as the material falls, so that it was better suited to an earlier day when the ore was richer and the mineral-grain coarser. A similar calciner, the White-Howell, was used in Canada to recover arsenic from auriferous mispickel. In Silesia and Saxony, where a good deal of arsenic is produced, the ordinary reverberatory roasting-furnace is used, both for the disengagement of the crude arsenic and the subsequent refining; in Cornwall, the reverberatory furnace is only used in the refining.

Exceptionally, calcining in Cornwall has been accomplished in multi-hearth furnaces, a circular furnace such as the Humboldt and a rectangular furnace such as the Merton. This use of multi-hearth furnaces in calcination is, however, unimportant compared with their ordinary use as roasters, that is to say, furnaces in which heat is applied primarily to effect a chemical change in preparation for the final metallurgical operation.

The Humboldt furnace consists of a vertical brickwork-cylinder enclosed in a sheet-iron casing, and supported on cast-iron columns (Fig. 415). By firebrick arches the interior of this cylinder is divided into five storeys, the bottom of each storey being a flatly-inclined hearth about 10 ft. in diameter. Over each hearth two radial rabble-arms are moved round the circle by a hollow vertical shaft at the centre. On these arms are rabbles so set that while those on the upper hearth plough the ore from the centre to the periphery where it drops to the second hearth, those on the second hearth work it to the centre again where it drops to the third hearth, and so on, till finally it is discharged at the periphery of the fifth. A separate firebox outside the cylinder furnishes any heat necessary, the hot gases being introduced at the fifth hearth, or it may be at the third. The hollow central shaft is driven by bevel gearing at a speed of about 1·25 revolutions per minute; it is cooled by cold air rising through it, this cooling extending to the hollow rabble-arms. Working in series with a separate rotary-drier and treating slime concentrate, this furnace in Cornwall had a capacity of about 8 tons per day; where used there was no arsenic to recover, calcination was preparatory to a simple treatment on tables and frames. The same roaster working on other and coarser material would run quicker and have a greater capacity.

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[1] *M. Mag.*, July 1910.

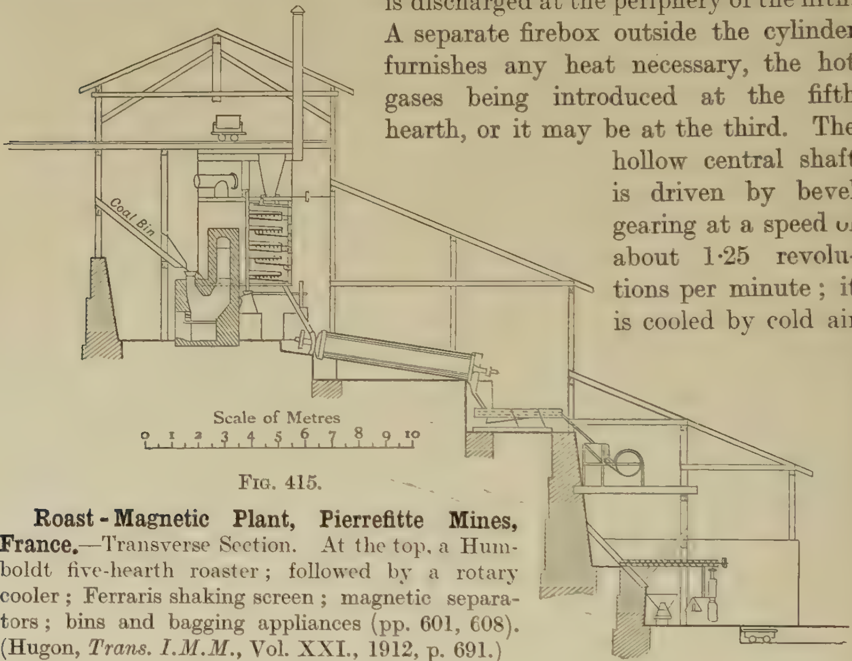

FIG. 415.

Roast-Magnetic Plant, Pierrefitte Mines, France.—Transverse Section. At the top, a Humboldt five-hearth roaster; followed by a rotary cooler; Ferraris shaking screen; magnetic separators; bins and bagging appliances (pp. 601, 608). (Hugon, *Trans. I.M.M.*, Vol. XXI., 1912, p. 691.)

The Merton furnace is a three-hearth rectangular brickwork-construction with two vertical spindles regularly disposed along the longitudinal axis, each spindle working a single rabble-arm on each hearth in such rotation and with rabbles so set that the ore is worked along the top hearth to drop at the end on to the second hearth, where it is worked back in the reverse direction to fall in due course upon the bottom hearth, at the end of which it is discharged (Fig. 416). Heated gases entering at the bottom from a firebox move in the opposite direction to the ore, finally leaving the furnace by a flue at the top near the feed hopper. The top of this furnace, like that of the Brunton calciner, constitutes a useful drying-floor. The rabble-arms and the hollow vertical shafts are cooled by a water circulation.

The application of heat to decrepitate one mineral of a mixed concentrate and thereafter to separate it as the undersize of a screen having appropriate apertures, affords another instance of disintegration by calcining.

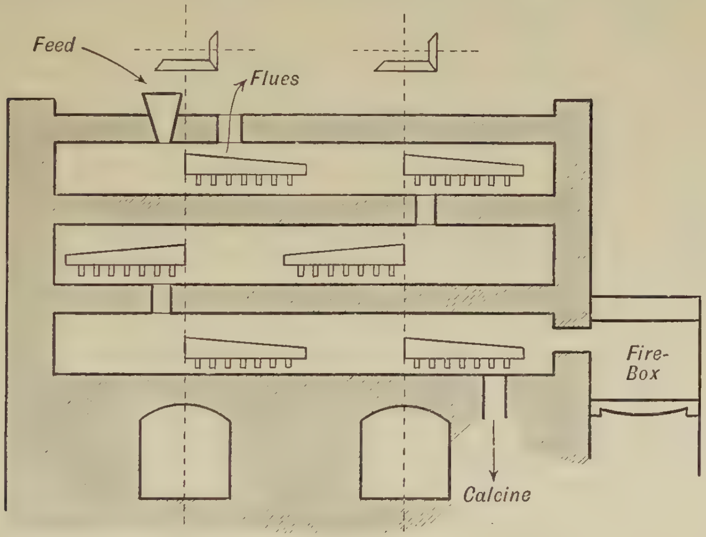

FIG. 416.

Merton Roaster.—Diagram. The rabble-arms and spindles are water-cooled (p. 602).

In the Missouri zinc district many extensive low-grade blende deposits contain much barite. These two minerals have so much the same density that they come together in the concentrate obtained by water. They may, however, then be separated by heating the concentrate to no great temperature, the barite decrepitating while the blende yet remains whole. At the Tahoma mine, where such a separation was practised, the concentrate was fed into an iron pipe 12 ft. long and 8 in. diameter supported in a brick furnace in a manner permitting its rotation. At the discharge end of this pipe and co-axial with it was a compound-oversize trommel having sieves of 20, 30, 40 and 50 mesh. Arrived at this trommel the blende became separated as the oversize, and the barite as the final undersize through 50 mesh, 90 per cent of this undersize being small enough to pass 200 mesh.

Only about 10 per cent of the blende went with the barite. The same procedure has been followed at a mine in southern Spain.

Another exceptional use of heat to secure release of a valuable mineral is in the liquation of stibnite, the most fusible of minerals, from the rock in which it occurs.

Finally, the release of metallic gold by calcining the pyrite, mispickel, etc., with which it occurs mechanically associated, has long been practised in many countries. With the pyrite thus disintegrated, the gold is readily recovered by panning, by amalgamation, by chlorination, or by cyanidation.

The calcining to disintegrate and release which has been described has been the application of heat treatment to the concentrate. It is, however, not impossible that such a treatment might be advantageously applied to crude ore. The disintegrating effect of heat upon ore has long been known; it was indeed applied in winning ore by fire-setting, where it was accompanied by quenching. No similar treatment of crude ore directly to assist disintegration and to favour release of the mineral has yet been adopted in practice, though attempts have been made to estimate the possible benefit.[1] It is known, however, that quartz, the most abundant of gangue-minerals, suffers important molecular changes under heat, such changes being mostly such as lessen its strength; at 575° C., for instance, 'sliding or gliding planes' so develop, that, after heating and quenching above that temperature, a quartz crystal can almost be twisted to pieces between the fingers. It is the heat and not any subsequent quenching which produces this condition; quenching, though it may introduce secondary cooling-strains, serves primarily to fix the condition reached by the heat. If the material could be crushed in the hot condition no quenching would be necessary. In the ceramic industry these 'transformation points' of siliceous materials have been closely studied with the view to their avoidance when baking pottery products; contrariwise, in ore-dressing the study would be to meet and use them.

Conceivably, with ore weakened by such heat treatment, the release of contained minerals would be the more readily effected, and with less damage to the mineral-grain. In this direction there are possibilities with granular argentite contained in hard chalcedonic quartz, with granular cassiterite in silica-cemented ore, etc.

Magnetic Roasting

Magnetic roasting or roasting for magnetism is that heat treatment undertaken purposely to create such magnetic properties as shall provide a basis for mineral separation; it is particularly applied when differences in mineral density do not suffice for separation.

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[1] Yates, *Trans. I.M.M.*, Vol. XXVIII., 1918, p. 41.

Miners were long ago aware that non-magnetic and feebly-magnetic iron minerals, such as pyrite, marcasite, mispickel, siderite, etc., could by roasting be rendered strongly magnetic, and of this knowledge advantage has long been taken in magnetic separation. They also realized that the magnetic properties of the resulting calcine varied widely with the conditions, oxidizing or reducing, under which the calcination had been conducted, the magnetic oxide preponderating under reducing conditions and the feebly-magnetic ferric oxide under strongly-oxidizing conditions. Nor did it escape their notice that by a quick roast pyrite might be converted into the magnetic sulphide. These are still the bases of magnetic roasting. Similar awakenings of magnetism are observable with minerals which though not purely iron minerals yet contain iron, for instance, with chalcopyrite, bornite, marmatite (ferruginous blende), etc.

In this matter the transformation points of pyrite are informative, not only because that mineral and its isomer, marcasite, are so frequently the minerals concerned, but also because the transformations of the more complex sulphides are not dissimilar. The decomposition of pyrite on roasting with excess of air may be taken to be approximately as follows: pyrite decrepitates at about 60° C.; at 400° C. it loses sulphur rapidly, becoming the magnetic sulphide, purple-black and iridescent, the sulphur igniting to sulphurous dioxide; at 500° C. the mass becomes incandescent, with the formation of the magnetic oxide, dull and dark in colour, the remaining sulphur burning to the dioxide; at the same temperature, after a time, the magnetic oxide by combining with additional oxygen becomes the feebly-magnetic ferric oxide, dull and red. Thus:

$$
\begin{aligned}
&7FeS_2 = Fe_7S_8 + 6S;\ \ 6S + 12O = 6SO_2\\
&3Fe_7S_8 + 76O = 7Fe_3O_4 + 24SO_2\\
&2Fe_3O_4 + O = 3Fe_2O_3.
\end{aligned}
$$

Roasted without excess of air, the formation of ferric oxide would be so delayed that it would be possible to stop the operation more or less at the magnetic oxide. Roasted in the presence of a reducing agent, finely-divided coal, for instance, the ferric oxide stage would not supervene, but, on the contrary, were any ferric oxide present it would be reduced to the magnetic oxide; thus:

$$6Fe_2O_3 + C = 4Fe_3O_4 + CO_2.$$

Pyrrhotite, the magnetic pyrite, first ignites at 500° C., the subsequent developments being the same as with pyrite.

The transformation points of siderite, ferrous carbonate, may be taken to be as follows: with abundant air and at 300° C., carbonic acid is expelled and oxygen absorbed, with the formation of ferric oxide thus:

$2FeCO_3 + O = Fe_2O_3 + 2CO_2$. With air in moderate amount the magnetic oxide is formed and the operation may be stopped before the formation of ferric oxide, thus: $3FeCO_3 + O = Fe_3O_4 + 3CO_2$.

Red haematite, as indicated above, is converted into the magnetic oxide by roasting in the presence of a reducing agent; the same is true of brown haematite, the water of hydration being first expelled.

In practice, magnetic roasting does not necessarily proceed to the complete transformation of the mass, but aims at the development of a magnetic film sufficiently deep that in an appropriate field the whole mass responds.

The most important present application of magnetic roasting is in the beneficiation of the marcasite-blende concentrate so frequently produced by zinc mines. In the Wisconsin zinc district of the United States this concentrate contains about 35 per cent of zinc and 25 per cent of iron, both combined with sulphur; such material cannot be retorted. By a magnetic-roast and subsequent magnetic-separation the zinc content is raised above 60 per cent while the iron is reduced to about 3 per cent. In that district the magnetic roast is conducted in seven-hearth furnaces about 22 ft. diameter and standing about 24 ft. above the floor; the top of the furnace in addition is a useful drying-hearth (Fig. 417).[1] Roasting proceeds by the burning of a portion of the contained sulphur; except at starting no outside fuel is required. Fed on to the drying hearth, the ore is rabbled to the centre to drop on to the first roasting-hearth; across this hearth it is moved to the periphery, whence it drops to the second hearth to be brought to the centre again; and so on, until the discharge takes place from the periphery of the seventh hearth; at this bottom hearth the temperature is highest, about 475°—525° C. The amount of ore roasted by such a furnace per day is roughly 125 tons. With the quick passage this tonnage connotes, and at the relatively low temperature obtaining, the marcasite, in spite of the abundance of air passing, is converted more or less to the magnetic sulphide. Evidence of this lies in the fact that the magnetic product subsequently obtained from the separator still contains sufficient sulphur to make this product valuable for the manufacture of sulphuric acid. The gases of combustion, carrying with them about 1·6 per cent of the ore, go through a dust chamber to a Cottrell electrostatic precipitator.

From the roasting furnace the ore is delivered to four rotary coolers, steel-plate cylinders 2 ft. inside diameter and 26 ft. long, cooled by a spray of water pouring directly upon them. No attempt is made to cool the ore completely, as the roasted ore has been found not to be so magnetic when cool as when somewhat warm.

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[1] Deutman, *E. & M.J.*, June 28, 1919, p. 1107.

On the basis of 125 tons of green ore roasted per day the cost of roasting and separation was about 5s. per ton, this figure not including general costs.

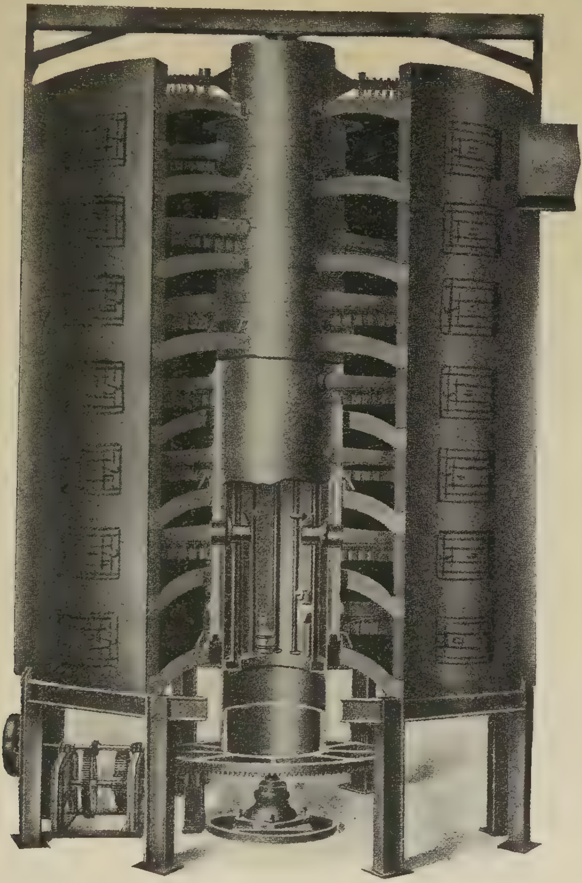

FIG. 417.

Wedge Roaster.—Part-sectional Elevation. The roaster shown has seven roasting-hearths and, on top, a drying-hearth. The hollow central shaft is of riveted steel plate lagged outside with fire-brick which revolves with it. The draught from bottom to top keeps this shaft cool so that a man can enter to attend to the rabble-arms. These arms revolve with the shaft 15—30 revolutions per hour; they communicate with a cool-air service on top. The capacity of such a furnace 22 feet diameter is ordinarily 75—100 tons per day (p. 606).

In Europe the association of marcasite with blende in ore-deposits is not so characteristic as in Wisconsin; on the other hand, it occasionally happens that ordinary pyrite is the contaminant of a blende concentrate, and in such circumstance both the practice and the means employed are very similar to that just described.

At the Pierrefitte mine, Hautes Pyrénées, France, water-concentration separated argentiferous galena and a pyrite-blende middling.[1] This middling, amounting to about 40 tons per day, was roasted in two Humboldt five-hearth mechanical furnaces of about 13 ft. outside diameter, the rabbles making 5 revolutions per minute; serving these two furnaces was a separate fire-box (Fig. 415). From them the roasted ore passed to a rotary cooler about 18 ft. long, 3 ft. 6 in. diameter, borne upon trunnions and set at an inclination of about 1 in 6. This cooler contained 16 water-jacketed tubes, each with 3 inches clear diameter for the passage of the ore and a ½ in. annular space for the passage of water. The capacity of this cooler was 40—50 tons per day, the provision of cooling water being about ten times this weight. The material to be roasted contained about 30 per cent of zinc and 16 per cent of iron, and the product after roasting and magnetic separation about 48 per cent of zinc. Apparently, roasting of the pyrite was fairly complete; ordinary fuel was employed, and a draught created by an exhaust fan; the bottom hearth was kept at a good cherry-red heat, while the top hearth was black and merely warm. The dust was deposited in large sheet-iron V-bottomed chambers. The operating cost of this treatment, roasting and magnetic separation included, was about 3s. per ton of material treated, and the recovery about 94 per cent.

The association of blende with siderite, more common in Europe, was formerly likewise broken by a roast-magnetic treatment, though now this separation of siderite is largely accomplished unroasted in fields of high intensity. It was common practice in Sardinia, for instance, to calcine the blende-siderite middling in Oxland calciners after mixing with about 2 hundredweight of fine coal per ton of middling. These calciners were 30—40 ft. long and 2 ft. diameter; they made 16—24 revolutions per hour and had a capacity of about 1·5—2·5 tons per hour. This practice still maintains where the zinc mineral, instead of being wholly blende, is largely an oxidized mineral which it is desirable to calcine.

The calcining of Cornish tin concentrate has already been described; such roasting was practised long before magnetic separators were introduced, primarily to release the cassiterite from attached sulphide, to disintegrate both the free and the attached sulphide, and to disengage the arsenic, so that by subsequent buddling and rebuddling the cassiterite would be obtained clean. The roasted iron is, however, magnetic, and more readily separable magnetically when the amount justifies this refinement; the

[1] Hugon, *Trans. I.M.M.*, Vol. XXI. p. 691, 1912.

restricted volume of air suitable to the deposition of the arsenic, the quiet roast necessary to minimize the loss of fine tin, both favour a product which, though sweet in respect to sulphur, is yet largely the magnetic rather than the ferric oxide.[1] Whatever the option in the absence of wolframite, when wolframite is present in commercial amount, not only is magnetic separation doubly justified but necessary, since with the iron removed in a weaker field, the wolfram is separately recoverable in a stronger field, leaving the cassiterite so much the cleaner. For the wolfram to be recovered clean, however, much free ferric oxide should not be present, this oxide passing into the same magnetic product as the wolframite; accordingly, a light roast makes for clean wolfram.

At Llallagua, Bolivia, a pyrite-cassiterite middling obtained from water concentration is roasted previously to a wet magnetic-separation. As this middling contains 25 per cent of sulphur and roasting is only carried to the magnetic sulphide or oxide, no extraneous fuel is required after once the furnace has started. Roasting is conducted in multi-hearth mechanical furnaces which deliver a product still containing 10—12 per cent of sulphur. Treated in the wet separator the magnetic product contains 50 per cent of iron and 22 per cent of sulphur, the remainder being largely oxygen, while the non-magnetic product contains about 2·5 per cent of iron and 2 per cent of sulphur. Normally the roasted material is black; when it inclines to be red the magnetic separator will do poor work, the tin concentrate will contain much iron.

Magnetic roasting for the beneficiation of iron ores is not yet practised. It has been tried upon lean haematite ores from the Mesabi Range, Minnesota; it was also included in the Goltra Process, applied at Waukon, Idaho, to the beneficiation of a clay-bed containing nodules of haematite. In this process the crude ore was dried in rotary kilns 150 feet long and 10 feet diameter, by a hot blast entered at the discharge end, not only to dry but to carry away the dust into which the clay fell. The granular material normally discharged was plunged into water to permit the sorting of the large pieces, whence the undersize entered magnetizing kilns wherein oil-fuel was burned to reduce the haematite to magnetite.

In a reducing atmosphere a temperature something less than 500° C. is sufficient to convert haematite to magnetite; such a temperature and such an atmosphere can be obtained by the consumption of an amount of coal equal to about 10 per cent of the weight of the ore. Limonite appears to be quite as readily converted as haematite, probably because though there is water to expel, that expulsion leaves the mineral more porous to the reducing gases. In either case cooling must take place out of contact with air, or oxygen would be re-absorbed and the magnetite in part would return to haematite. It is considered that a magnetic-roasting of haematite, carried to the extent of converting 95 per cent to magnetite, would cost on a large scale about 4s. per ton.

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[1] Dietzsch, *Trans. I.M.M.*, Vol. XV., 1905, p. 2.

Though in this description of magnetic-roasting certain types of roasting furnaces have been mentioned, any modern type of roaster could be employed; much lies in the proper control of the air, much also in the duration of the operation. Roasting of pyrite to the magnetic sulphide need take little time if the temperature be sufficiently high; so conducted it is sometimes described as "flash roasting." Such roasting is characteristically conducted in a shaft furnace. Fed at the top of such a furnace, the ore falls to one side of a central diaphragm, through heated gases which, having risen from a firebox on the other side of the diaphragm, are now descending (Fig. 418). In those gases ignition takes place, disengagement of the sulphur continuing in the pocket into which the material falls, and from which by screw-conveyor the roasted material is eventually removed. Flash roasting is, however, liable to be uncertain and uneven.

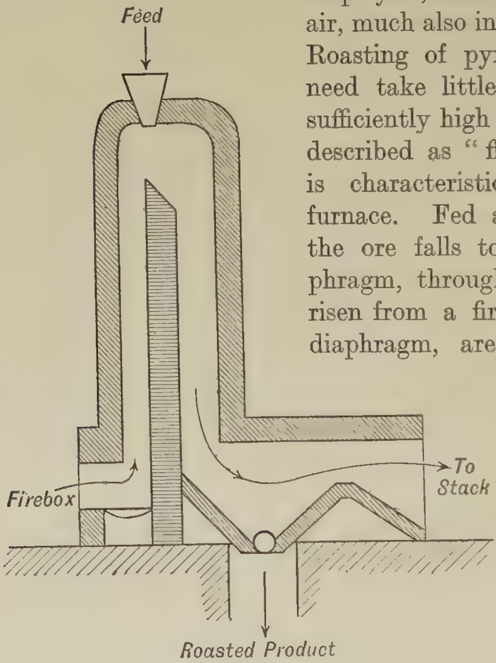

FIG. 418.

Shaft Furnace.—Diagram. Furnaces of this type have been used for the 'flash' magnetic-roasting of pyrite (p. 610).

Generally, when no feebly-magnetic mineral is present to be recovered separately but only iron oxide has to be removed, it is better to proceed to complete expulsion of the sulphur, because there is then the assurance that all the iron has been rendered magnetic. That some of the iron by over-roasting may have been rendered feebly-magnetic is a relatively small matter, since the energy consumed in the excitation of even the intense field necessary for the removal of such material is relatively low; in addition, fuel for the roasting is often largely available in the ore itself. Where such considerations apply, a good temperature and abundant air will not be feared but used to quicken the roasting.

FRACTIONAL ROASTING

Galena is more readily oxidized to the sulphate than blende; upon roasting its conversion to the sulphate takes place at a lower temperature; it is possible therefore to roast a mixture of these two sulphides at such a temperature that the galena is 'sulphatized' while the blende remains unaltered; this fractional roasting is more readily produced when the minerals are in a fine condition; a sulphatized surface is dull and so profoundly different from the lustrous metallic surface of the sulphide, that, instead of possessing any tendency to float, the mineral behaves like stony material and sinks. These phenomena are the bases of the Horwood process of differential flotation (p. 444).

In the Horwood process as applied at Broken Hill the mixed-sulphide slime is roasted at a temperature of about 400° C. with free rabbling and abundant air, the galena surface being thereby rendered dull and earthy while that of the blende remains bright and lustrous. With this alteration accomplished, the blende floats in an ordinary flotation-cell and is recoverable with the froth, while the galena sinks. The degree of sulphatizing necessary to the commercial success of the operation varies with the size of the mineral grain; with fine material 70 per cent or so of the galena will be found to have been converted to sulphate; with coarser material a much smaller percentage, since in relation to mass the extent of surface is smaller.

In practice, roasting is conducted at a temperature of about 400° C., by heat supplied partly from a firebox, partly also from the combustion of the sulphur.[1] Except there be pyrite present, the temperature should not be allowed to rise beyond 450° C., or there will be needless risk of the blende becoming involved in oxidation; pure blende by itself does not, however, ignite below a temperature of 550° C. With pyrite present the temperature may be allowed to climb to 500° C. without risk, since this mineral appears to draw the forces of oxidation upon itself; indeed with pyrite present the temperature should be kept high, because until all that mineral is oxidized the sulphatizing of the galena is delayed. Pyrite is oxidized rather than sulphatized.

A similar fractional-roasting was tried at the Afterthought mine, California, in an endeavour to separate chalcopyrite and blende obtained together in a collective flotation-concentrate (p. 445). This concentrate was roasted in a 25 ft. nine-hearth Wedge mechanical furnace, wherein the temperature of the fired hearth was kept at about 475° C., the higher and the lower hearths being cooler, the bottom hearth having a temperature of about 200° C. At these temperatures the chalcopyrite was largely converted to oxide but some to sulphate; so long as the amount of sulphate formed did not represent too great a loss, its presence in the solution was beneficial to the subsequent preferential-flotation of the blende while keeping down the copper. The properly-roasted material if left to cool gradually presented a characteristic brown colour, but if cooled rapidly it was black and the copper tended to float. Proper cooling was therefore necessary; this was accomplished by a special cooler in which the material remained for about 10 minutes.[1]

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[1] Clark, *Min. Mag.*, January 1910.

The Horwood process was also tried in British Columbia, to separate blende from a middling in which it was associated with pyrrhotite and mispickel. Here fractional roasting was best conducted at 350—375° C., this lower temperature being necessary because the blende was largely in the form of marmatite, which at a higher temperature would suffer oxidation and loss.[2]

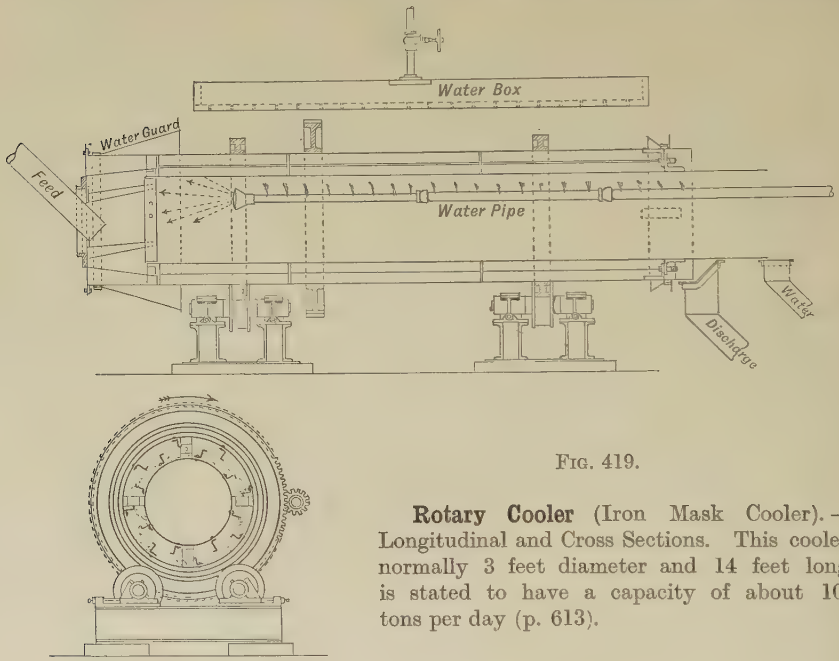

Fig. 419.

Rotary Cooler (Iron Mask Cooler).—Longitudinal and Cross Sections. This cooler, normally 3 feet diameter and 14 feet long, is stated to have a capacity of about 100 tons per day (p. 613).

Cooling

More than once in describing the various heat-treatments cooling has been mentioned (pp. 598, 606, 608, 612). Drying in rotary kilns where the ore moves away from the fire does not necessitate cooling, but when the

[1] Heller, *M. & S.P.*, Aug. 1919, p. 154.

[2] Motherwell, *M. & S.P.*, Nov. 29, 1919, p. 769.

ore moves towards the fire some sort of cooler may be required. Cooling is generally required with calcining because the ore moves towards the flame and, as it leaves the calciner, is quite hot. Special coolers are more necessary when the quantities are great; when the quantities are small the material can be spread out to cool. In many multi-hearth roasting furnaces the lowest hearth is a cooling hearth, rabbling taking place in air, which thereby becomes heated before entering the main combustion zone.

In design, coolers as a rule are very much like rotary driers, with water passing instead of hot gases (Fig. 419).

CHAPTER XIV

THE CONTROL OF OPERATIONS

Sampling and Assay Value; Weighing and Tonnage; Recovery and Enrichment

Whether of the separate operations embraced within a complete dressing-scheme or of the complete scheme itself, control is only possible when comparison in respect to quality, amount, and total valuable content, can be made between the original material and its products. It therefore becomes necessary to know the assay-values and weights of the material treated and of the products obtained, factors obtainable respectively by sampling and weighing.

Sampling and Assay Value

A sample is a conveniently small representative, in the commercial sense, of a large and given mass. In dressing, the material to be sampled is crushed ore in continuous or interrupted movement through the plant, and the given mass is that which passes in a given time. If ore were homogeneous any piece or portion would be a sample, the taking of which would present no difficulty; being usually, however, heterogeneous, fluctuating and capricious, the sampling of ore demands the most attentive and loyal care, and generally the assistance of machines designed to work impartially.

Samples collected at random never suffice. In the simplest case, say that of iron ore, pieces taken by the hand in accordance with a regular procedure may together make a reliable sample. More usually all the ore passes through a sampling machine, the sample taken passing in turn through other samplers in series till a proper weight remains. Accordingly, sampling may be conveniently described under hand sampling and machine sampling.

Hand Sampling.

Hand sampling is taken to include all sampling where the hand alone, or the hand manipulating a tool, is the active agency. As such it is described under the following headings :—

Truck Sampling.—Where the material passes in trucks, samples may be accumulated by taking, generally with the help of a scoop or shovel, a small portion from each truck. The reliability of such samples depends primarily upon the size of the particular material. With sand, for instance, and particularly when by previous crushing such sand has become thoroughly mixed, the results are reliable. A weight of 1—2 lb. is taken from each truck containing 1—2 tons, these portions being assembled to make a daily sample, which subsequently is cut down by laboratory means to a convenient weight; each sample-portion is therefore about 1-2000th of the material treated. Since a weight of 100—200 lb. is as high as is conveniently handled in the laboratory, the number of portions making up the daily sample will generally be about 100.

On the other hand, with broken ore proceeding from the breaker to the mill-bins, truck sampling is employed with little satisfaction, the range of size of this material being too great.

Car Sampling.—In the United States, iron ore is sometimes sampled by taking pieces regularly over the surface of the ore as it lies in railroad cars on its way to the furnaces. These cars, as a rule, contain 40—50 tons, from which amount about 20 pieces are lifted each at a determined point, regular spacing being obtained by means of a rope-net template. The ore being of relatively uniform value and broken to furnace size, this procedure is satisfactory and agreed between buyer and seller.

Tank and Bin Sampling.—Pulverized ore collected in tanks for such treatment as cyanidation may be sampled by pipes of design recalling the familiar cheese-triers. Such pipes are usually 2 inches in diameter and in length somewhat longer than the depth of the collected sand, 8—10 feet being common. At the top a tubular T-piece secures a handle by which the pipe is gradually worked down into the sand, and eventually withdrawn; at the bottom the rim is fashioned into a cutting edge; upwards from the bottom a slit extends nearly to the top, this slit giving a certain grip and springiness to the cutter, while permitting the ready entry of the sample (Fig. 420). When the pipe is withdrawn this sample is tapped out into a convenient receptacle. In an ordinary tank 10 to 20 such pipe samples are put down, all in positions respectively to represent equal portions of the tank tonnage. If the tank be rectangular they are spaced regularly over the surface; if it be circular they are aligned along radii, the holes gradually getting closer with distance from the centre. In either case a rope template with knots at the proper points, assists greatly in setting-out the pre-determined positions.

With tank sampling the number of holes in relation to the tonnage will depend upon the quality and character of the ore. Generally 100 lb. from 100 tons, a ratio of 1 in 2000, would be a fair sample-weight.

Concentrate collected in rectangular bins or in railroad cars may likewise be sampled by pipes. With such material, however, the pipe is smaller, say $1\frac{1}{4}$ in. diameter, and shorter, the depth of collected material being less; moreover, since concentrate packs and is heavy, the pipe has an iron or wooden plug on top, so that it may be beaten down through the collected material (Fig. 420). Sometimes, instead of a simple pipe an augur is used, around which, when screwed down, a pipe is beaten, augur, sheath, and sample being then lifted together.

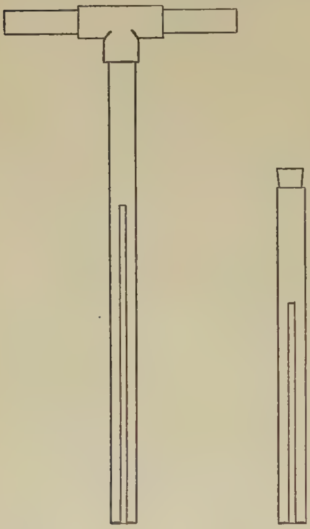

FIG. 420.

Pipe Samplers. — General Outlines. The sampler with the cross handle is for sampling sands; that plugged at the top is for sampling concentrate (pp. 615, 616).

Mortar-box Sampling.—The pulp from stamp-crushing, after issuing through the screens, flows over a lip which is part of the mortar-box casting, to drop into launders or on to amalgamated copper plates. Inserting at this drop a miniature trough, about 3 in. in section and the length of the mortar-box, the pulp issuing for about 3 seconds may be caught as a sample portion, nothing overflowing. Repeating this procedure every hour, the whole amount collected per day, being about 1-1200th of the material crushed, becomes the daily sample. Handles suitably disposed permit the sampler to be conveniently held in position, and afterwards its contents to be poured into the sample tank or drum (Fig. 421).

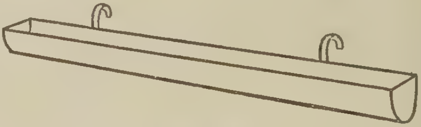

FIG. 421.

Mortar-box Sampler.—General Outline. The length of this miniature trough is the same as that of the mortar-box lip (p. 616).

Mortar-box sampling is largely practised on the Rand and elsewhere. When loyally carried out the samples obtained are reliable, particularly when the crushing is fine. Being, however, in the hands of men whose first duty is to attend to the stamps, the regularity with which the samples are taken often leaves much to be desired. Taken every hour the daily sample is an accumulation of 24 contributions. Care must be taken that the sampler is not filled to overflowing, and that at the end of the day only clear water is decanted.

Filter-cake Sampling.—Filter-cake material, even though it may be dry enough to transport on belt-conveyors, is yet generally so sticky as to preclude the use of a mechanical sampler. The slime of which such cakes consist may, it is true, be sampled mechanically before filtration, and while yet in the condition of pulp, since with freely-flowing pulp uncleanliness of the cutter does not develop. But in the absence of such mechanical sampling, and also when treatment takes place in the interval between the thin and the filtered conditions, it may be necessary to sample the filter-cake. Such sampling is done by taking pieces by hand at regular intervals.

Heap Sampling.—Heaps of ore or concentrate may be sampled by (*a*) fractional shovelling, (*b*) coning-and-quartering, (*c*) riffling.

"Fractional shovelling" consists in making two heaps with the shovel, one of these new heaps, the sample heap, being the accumulation of alternate shovelfuls, or of every *n*th shovelful. Where the mass of the first sample-heap is too large, this heap itself, after further crushing if need be, is submitted to a second operation, and so on. When the heap becomes or is small, and the material has been crushed fine or is fine, the shovel gives place to a relatively small scoop. However large the heap, fractional shovelling can only be considered for material which has passed the ordinary breaker, while for small heaps the fine breaker and, may be, the laboratory breaker, must be used. On the other hand, it cannot be practised with very fine material because of the dust which would arise. Granted a proper material, the procedure, honestly carried out, ensures reliability by the large number of cuts involved in the production of the final sample; however, it is laborious, and the continual necessity to remember to which heap to turn, is a strain. No special mixing of the heap is necessary, but in the piece-meal attack and reconstitution considerable mixing obtains.

"Coning-and-quartering" consists in building the ore into a cone by additions at the rising apex; when complete this cone is carefully flattened by pressure applied vertically downwards at the apex; the resulting flat circular bed is then divided into four quadrants, two alternates of which are carefully and completely rejected, leaving the remaining two to be built into a second cone, which in turn is quartered. This procedure continues, with any necessary reduction in particle-size, till the two remaining quadrants are taken as duplicate samples.

With coning-and-quartering care must be taken that the cone is built symmetrically, the larger pieces rolling downwards equally all around the circle, the finer material remaining at the centre ; no mixing in the ordinary sense takes place or is necessary. Similar care must be taken that, in the process of flattening, the rich central apex is not forced into any one quadrant, but partly and equally into all. With these attentions the results obtained, in spite of the small number of cuts entailed, are reliable, and the procedure is largely practised, if not from start to finish, at least in the last stages of a sampling operation. It is laborious ; but, since the cut is made when the hurly-burly of shovelling has ceased, no mental strain is experienced in keeping 'sample' and 'reject' in their proper and separate paths.

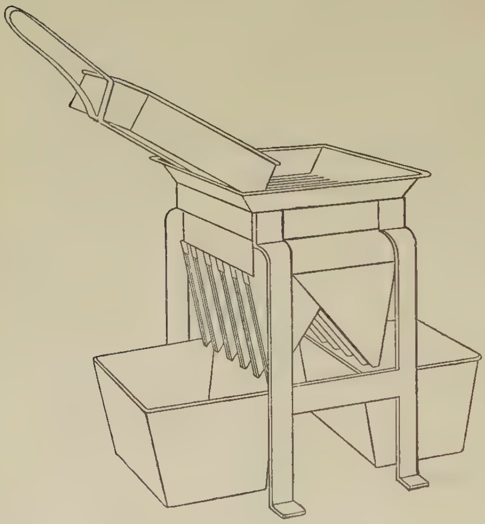

Fig. 422.

Jones Riffler.—Perspective View. This view of a small riffler well shows six spouts leading to one side and six to the other, with two pans into which the products respectively fall (p. 618).

"Riffling" consists in pouring the suitably crushed material on to a grid so constructed that the portion falling on the bars may be collected separately from that falling into the spaces (Fig. 422). The bars of this grid are not solid, but are narrow chutes with inclined bottoms ; similarly, the spaces between the bars are not bottomless, but have an inclined bottom ; hence, other than that the bottoms are inclined oppositely, the bars and spaces are identical in figure ; they are also of the same dimension. Generally, six bars and six spaces make the complete riffler, so that the two halves into which the material is divided are the result of twelve cuts. Such rifflers, known as Jones rifflers, are not used with large material ; the upper limit of size is about an inch, this size demanding chutes about two inches wide. Larger than this, the appliance would become unwieldy, and the manipulation of the pans into which the products fall would be difficult. In the laboratory, small rifflers are much used.

Time Sampling.—Samples taken at any moment around a concentrating machine in operation, of the feed and successively of the products, are illuminating in the light they throw upon the work of the particular machine. Though such samples are cut from continuous streams, mechanical sampling is out of place, because the information desired is not such as to warrant the complication of a continuous record, being of too internal and detailed a character. Moreover, the hand sampling of such streams permits each to be sampled for a time suitable to its particular volume, so that in addition to securing samples for assay, indexes to the respective tonnages, treated or produced, are obtained, and the complete account of the work done by the machine may be closed ; hence the description of these samples as 'time samples.'

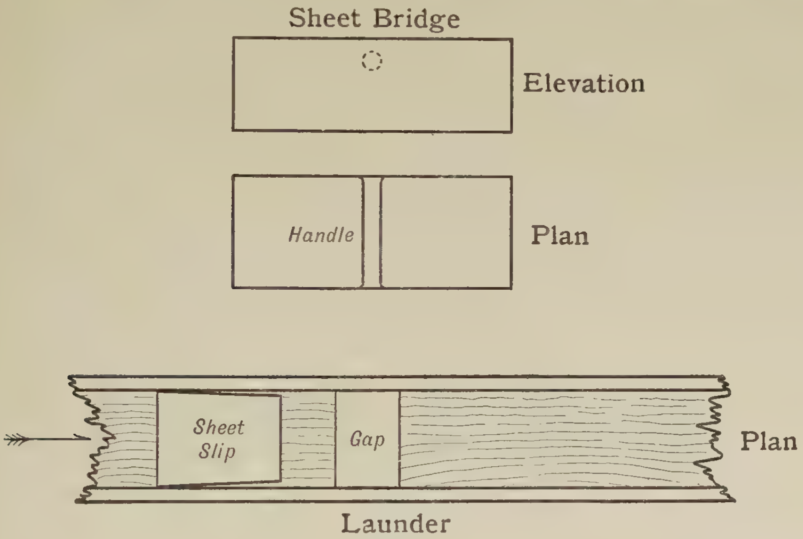

Fig. 423.

Stream Diversion for Time Sampling.—Sketches. At an appropriate spot a gap is left in the bottom of a launder, down which gap, if open, the stream would drop. Carrying the stream over this gap is a sheet-bridge provided with a handle by which it can be withdrawn downstream to open the gap, and be brought back into position after the time-interval. Though this bridge fits the launder closely, tightness of the joint is further secured by bringing the bridge upstream behind a sheet-slip fixed in the launder (p. 619).

Time sampling consists in diverting and collecting in a suitable vessel the whole of a particular stream for a given time (Fig. 423). This time depends upon the size of the stream ; it may be anything from 15 seconds to 150 seconds. A bucket or sheet-iron drum with appropriate handles makes a convenient vessel. In that vessel the pulp is allowed to settle till the water can be decanted clear, the amount decanted being measured ; the remaining dense pulp is then carefully dried. In the end, therefore, both the weight of the water and that of the dried sample are available.

In diverting the stream care must be taken that any accumulations upstream are not disturbed, and that no undue portion of the heavier material moving along the bottom empties itself into the sample. With these precautions taken, time samples are of great help in the intelligent control of concentrating machines. As will be described later, they are sometimes used in broader issues, namely, in the determination of mill tonnages (pp. 630, 634).

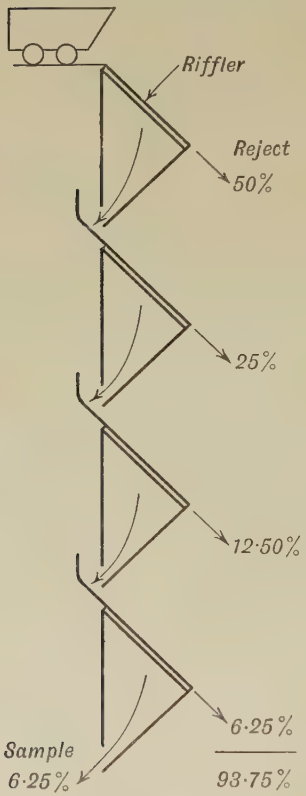

Fig. 424.

Fixed-cutter Sampler.—Diagram. Rifflers in stepped series are indicated (p. 620).

Mechanical Sampling.

Unlike hand sampling, mechanical sampling demands the ore to be moving in a continuous stream. In sampling such a stream the procedure may be considered in relation to two and opposite principles, namely, either *one and the same portion of the stream may be taken the whole time*; or, *the whole stream may be taken a portion of the time.*

Under the first principle, a fixed portion of the stream taken the whole time, it is obvious that for that fixed portion to have any claim to be representative it must be substantial, say one-half ; accordingly, there must be opportunity for repeated and similar subdivision. Further, the distribution of the ore across the stream must be uniform and the material loose, conditions which are not fulfilled by any water-borne stream nor by any stream moving upon a flat bottom. Accordingly, this principle of sampling can only be applied to a falling stream of dry ore, and by fixed or stationary cutters. Mechanical samplers with "fixed cutters" are but a development of rifflers; they differ from ordinary mechanical samplers in having no moving parts, and from rifflers in requiring no handling ; gravity is the active force and height the necessity ; cut follows cut in succession, till finally a sample of convenient size issues at the bottom (Fig. 424). With no opportunity for intermediate crushing, such samplers are only suitable for relatively fine ore. They are little used; that known as the "whistle-pipe sampler" is representative of the type (Fig. 425).

The second principle, namely, the whole stream taken a portion of the time, is that applied in time-sampling by hand, already described. There, however, it was applied, not to obtain a representative sample of the mass passing during a long period, say a day, but to obtain samples representative of the material passing for a short period extending equally to either side of the actual time of diversion. Where the stream is small, as would be those around any single concentrating machine, the necessary diversion of the whole stream is possible without splash or scatter, but the similar diversion of a main stream is not possible with precision. Accordingly, in the sampling of a main stream, the stream itself is not diverted, but at regular intervals a cutter moves across it, this cutter receiving the stream into itself to lead it away.

Though the edges of such a cutter are aligned with the stream, the movement across the stream may not be too rapid, or there will be splashing and irregularity; on the other hand, if the cutter moves slowly and is large enough to take the full stream, an inconveniently large sample is taken. This position is met by dimensioning the cutter not to take the full stream at any one time, but only a fraction of the stream; that is to say, *instead of taking the whole stream a portion of the time, it is usual to take each portion of the stream in succession a portion of the time.*

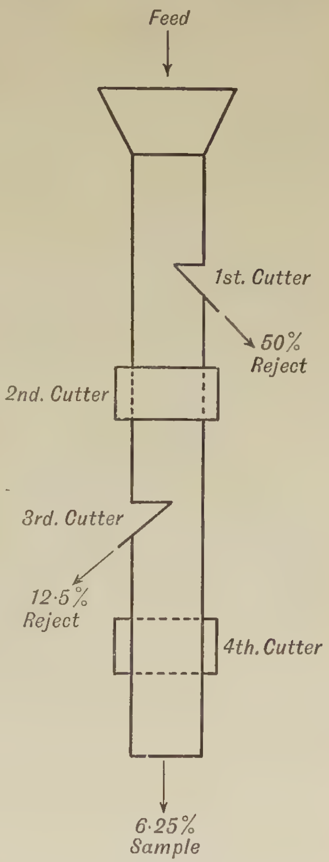

Fig. 425.

Whistle-pipe Sampler.—Diagram. In this sampler the fixed cutters are inclined blades inserted half-way into the pipe down which the stream of ore is falling. Each cutter in succession rejects one half of the stream falling upon it, till eventually the bulk is reduced to that of a convenient sample. Successive cutters are arranged 90° ahead of one another round the pipe circumference (p. 621).

The precise relation between the width of the stream and that of the cutter will depend upon the speed of movement and the weight of sample necessary to make it representative; beyond this, the cutter must have such dimensions that the largest pieces enter readily. In so far as chance affects the accuracy of practical sampling, according to theory the error of any particular sample diminishes as the square root of its weight increases, and the error of the average of a number of samples equally diminishes with increase in the square root of the number of samples. Both of these errors must be reasonably small; the individual sample must be reasonably representative of the amount passing at the moment; and the number of samples must be such as to make the average reasonably representative of the whole mass passing during the particular period. These two factors are, however, not on the same plane of importance; perfect correctness of an individual sample cannot be expected to diminish error of the average due to insufficiency in the number entering that average; but a large number of samples can be relied upon to mitigate, if not to eliminate the effect of the error of the individual sample. Accordingly, in sampling, it is rational to accumulate an average sample by a relatively large number of cuts, each of relatively small weight, rather than by a small number of cuts of large weight. More important still, experience teaches that the number of cuts must always be relatively high, even though, as happens particularly when sampling coarse ore, the large bulk of the resultant sample necessitates reduction by further sampling in series.

Mechanical samplers, bending to the force of the foregoing considerations, are now exclusively "moving-cutter samplers." These, since the stream of ore to which they are applied may either be dry or water-borne, it is convenient to describe under dry-stream samplers and pulp samplers, respectively.

Dry-stream Samplers.—Sampling of the crude ore must be undertaken before the ore is divided or anything taken from it. All other things being equal, the finer the material the more reliable the sample. The introduction of water destroys the equality of condition between dry material and water-borne pulp. By many it is considered sampling had better be undertaken before wet crushing begins; certainly, the sampling of dry ore properly carried out is sufficiently reliable to form the basis of purchase and sale. Any dry stream of ore is generally of coarse ore, say the product of breakers. With such material certain samplers have shown themselves reliable in result and in mechanism.

The Vezin sampler usually consists of two hollow cone-frustra placed base to base with their common axis vertical (Fig. 426). Continuing this double cone above and below, are sections of a hollow pipe which serves as the shaft by which the cone is supported and rotated, though in some designs the double cone is suppressed, the hollow shaft being continuous. Emerging from the side of the upper of the two frustra is a radial cutter, with vertical sides and horizontal cutting-edges. Rotation of this cone and cutter takes place within a cylindrical casing, with a sloping bottom pierced by the hollow shaft. The stream to be sampled enters this casing down an inclined chute, which delivers it over the cutter, tangentially to and in the direction of revolution. Rotating under the stream, the cutter at each revolution takes of the ore passing during that revolution, a proportion determined by the ratio of the angular magnitude of the cutter to the complete circle. This contribution to the sample enters the hollow frustra and is discharged at the bottom of the hollow shaft, to accumulate till the day or shift is done.

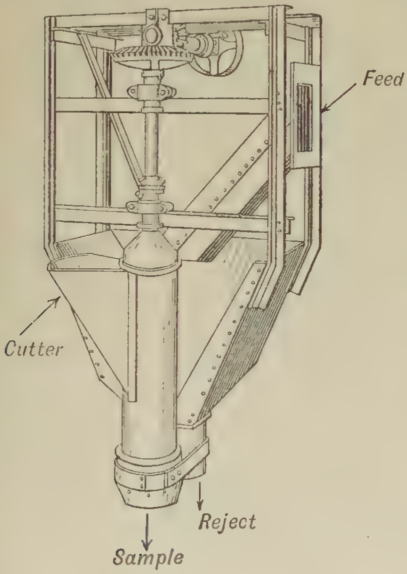

FIG. 426.

Vezin Sampler. — General View. The type illustrated shows the radial cutter standing out from a hollow pipe and not a hollow cone; nor does the casing contain the cutter throughout its revolution, but only through the segment traversed by the ore-stream (p. 622).

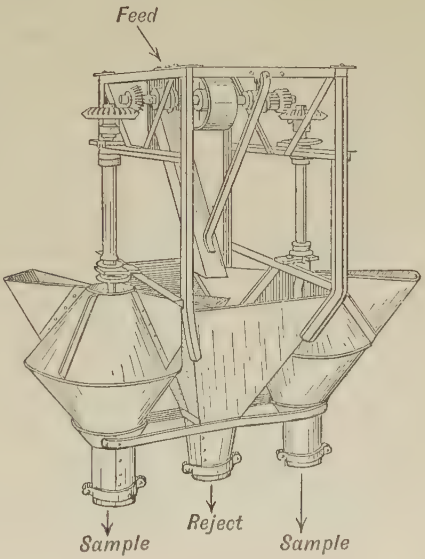

FIG. 427.

Vezin Duplicate Sampler. — General View. The cutters of two samplers pass successively through the same stream of ore, each taking an independent sample, while the common reject is assembled by a pyramidal casing between them. In the illustration the cutters are seen standing out from hollow cones, this being the standard design (pp. 622, 624).

The casing is generally about 2 feet diameter; the cutter is wide enough readily to take 2 in. pieces; the angle between the radial sides is generally 18°, so that one-twentieth of the ore, that is 5 per cent, is taken as sample, this proportion being independent of the speed of rotation, which is about 20 revolutions per minute. At such a speed the material falling down the chute drops fairly dead into the cutter; a higher speed would cause scattering of the material and damage to the cutting edges. Commonly, two cutters are placed diametrically opposite one another, in which case 10 per cent of the ore is taken instead of 5 per cent, and the machine is better balanced; exceptionally, two single-cutter samplers follow one another under the same stream to take duplicate samples (Fig. 427).

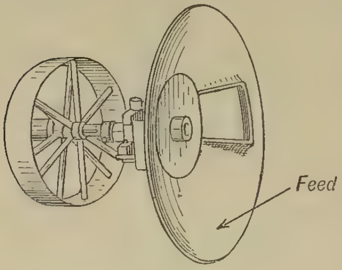

Fig. 428.

Snyder Sampler. — General View. A rotating pan intercepts and deflects the stream of ore, the sample passing through a hole in the periphery (p. 624).

The Snyder sampler consists of a saucer-shaped pan about 3 feet diameter, rotated at a speed of about 30 revolutions per minute in a vertical plane by a horizontal shaft carrying the driving pulley (Fig. 428). Into the concavity of this pan an incline chute leads the ore in a manner to strike the pan near its periphery, and, striking there, to be deflected backwards. At one point in the path of impingement a hole permits a portion of the ore to pass clean through. The portion so passing is the sample, the proportion of which is determined by the relation between the width of the hole and the length of the complete circular path; ordinarily it is 5—10 per cent of the ore.

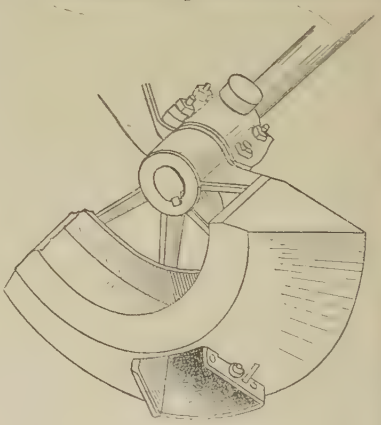

Fig. 429.

Brunton Sampler.—General View of the Cutter. A cutter oscillates in a falling stream, deflecting all the stream forward, except for a fraction which it deflects backward (p. 624).

The Brunton sampler is similar to the Snyder, but the pan, instead of being whole, is only a minor segment of the complete circle, and the movement, instead of rotation, is one of oscillation (Fig. 429). By such a design the cutter, after making its cut, has not to race round the circle before making the next cut, but moves only through the segmental arc, say 120°; hence, for the same number of cuts made per minute the peripheral speed is much slower, and irregularity by scattering is diminished. Moreover, this sampler, on account of its small vertical dimensions, may be inserted where other types would not find space. On the other hand, an oscillatory motion is neither so simply produced nor maintained as that of rotation.

The foregoing samplers are standard machines readily obtainable from machinery makers; provided they are in good order, the samples taken are accepted as reliable. They are largely employed in sampling ore for purposes of sale; in sampling ore undergoing treatment in a custom mill or concentrator belonging to others; and in sampling ore at the smelter. For such work the relatively large sample taken by these samplers is warranted.

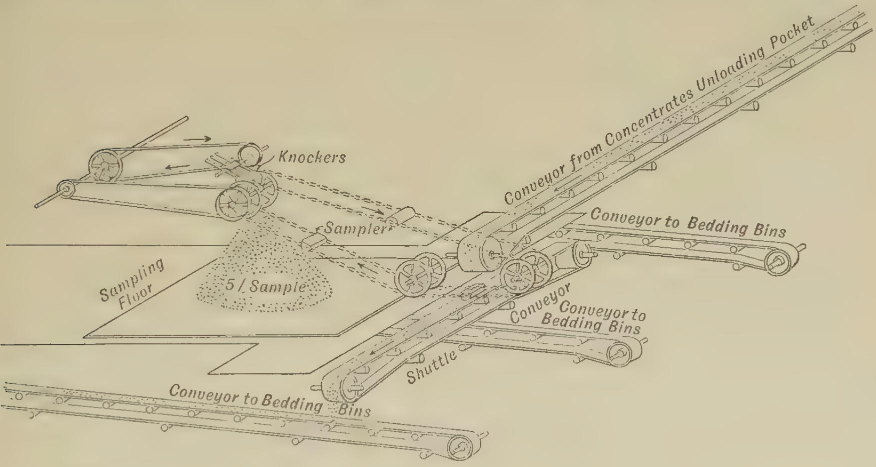

Fig. 430.

Bucket-conveyor Sampler.—Diagram. An arrangement for sampling concentrates at the smelting plants of the New-Copper Company, Arizona, is shown (p. 626). (MacGregor, *Trans. A.I.M.E.*, Vol. LV., 1917, p. 783.)

With domestic mill-ore the quantities involved are larger, and the values lower and more regular, conditions under which a smaller proportion will suffice as sample; moreover, the samples being taken for domestic control and not with a view to wide acceptance, samplers designed by the domestic staff may compete with those of standard design.

It is common to see mill-ore sampled as it drops from the end of the conveyor bringing it from the breaker station. The cutter under such circumstances may be a bucket which, moving horizontally round a circle, cuts the stream, taking as sample a proportion determined by the ratio between the dimension of the bucket, measured along its horizontal path, and the length of the complete circular path. Ordinarily, this proportion is less than one per cent; sweeping around a circle of 12 feet diameter, for instance, and with a bucket width of four inches, it would be about a one-hundred-and-twentieth. Such buckets have a bottom discharge, which at an appropriate place is tripped open and then shut again. Or the bucket, instead of moving in a horizontal circle, may be carried upon two endless chains, these chains running around pulleys so placed that as the bucket cuts through the stream it fills, tipping taking place afterwards (Fig. 430). Sometimes, again, a sample is taken by omitting one plate of a belt feeder, the sample then falling into a pan beneath, its proportion depending upon the number of plates making up the complete belt.

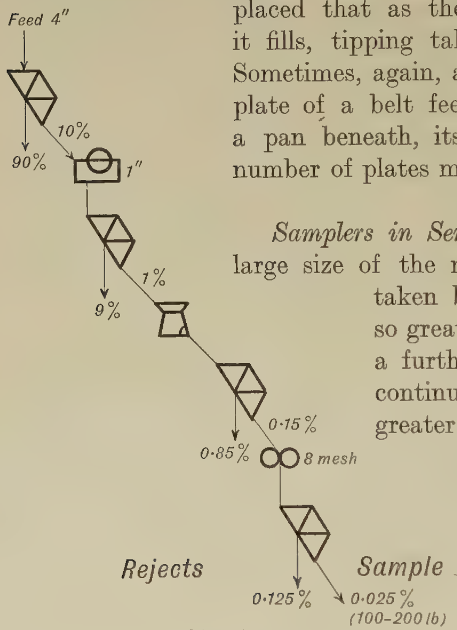

FIG. 431.

Sampling Equipment at Cananea, Mexico.—Flow-sheet. This sampling equipment is for domestic copper ore. The final sample (100–200 lb.) is mixed and riffled to 4 lb.; then, after drying and grinding, to 2 lb., this weight after regrinding to 200 mesh being finally divided into 4 lots each of half a pound (p. 626).

Samplers in Series.—On account of the relatively large size of the material, the accumulated sample taken by dry-stream samplers is usually so great as to necessitate its submission to a further sampling, this procedure being continued till the amount remaining is not greater than can be handled by laboratory means. To maintain the accuracy of the original sample through such a reduction in amount, the sample must be re-crushed before such further sampling; accordingly, the practice in sampling is for crushing machines and samplers to follow one another more or less regularly (Fig. 431). Experience indicates that the rate of reduction in bulk should not be faster than the rate of reduction in particle-size. Should the reduction in particle-size not be so fast as that in the sample-mass, then to maintain the accuracy the proportion taken as sample must increase (Fig. 432); on the other hand, if particle-size is reduced the faster, the next cut may be smaller.

It might even be that the crushing at one stage accomplished a size-reduction sufficient for two successive samplers, with considerable simplification in equipment. It would then, however, be advisable to interpose a mixing drum between the two samplers, in order that the second sampler should have a continuous feed. To the same end the original sample of such a series is generally delivered into a pocket from which it is regularly drawn by a feeder at the head of the sampling equipment.

Consisting of several stages down through which the respective rejects and samples usually fall by gravity, the sampling equipment requires considerable height. The ultimate sample delivered by this equipment will generally be about 100 lb. in weight. The tonnage represented by this sample, and the fineness to which this sample is brought, will depend upon the nature of the ore. With domestic low-grade copper ores, for instance, a weight of 100 lb. of material crushed to $\frac{1}{4}''$—$\frac{1}{2}''$ might well represent the daily tonnage entering the mill, even a thousand tons. With custom gold-ores, on the other hand, two duplicate samples each of 100 lb. and crushed to 1 mm. might be required to represent 100 tons, particularly if the ores were spotty in character. Accordingly, the proportion which the sample delivered by the mill sampling-equipment bears to the tonnage sampled varies from 1-2000th to 1-20,000th.

In the laboratory the weight and particle-size of this sample are still further reduced, till eventually two duplicate samples each of about half a pound, crushed smaller than one-eighth of an inch when of low-grade ore and smaller than 80 mesh when of high-grade ore, remain for the assayer.

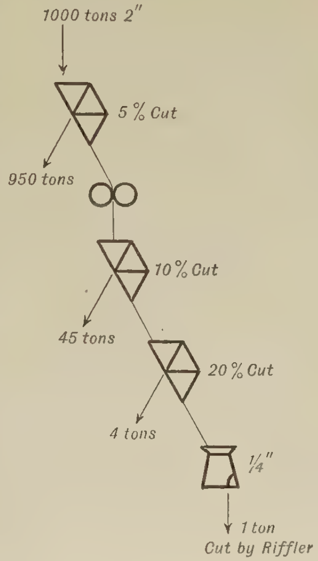

FIG. 432.

Sampling Equipment at Santa Gertrudis, Mexico.—Flow-sheet. The ore here is a siliceous silver ore (p. 626).

Pulp Samplers.—The samplers employed with pulp streams do not differ in principle from those used in the sampling of dry ore, nor does any hard-and-fast line exist between the two types. The material being very much finer, and generally in the condition of sand or sline, the cutter may be much narrower, taking a justifiably smaller sample.

Ordinarily, the cutter moves horizontally across the stream, taking each vertical portion in succession (Fig. 433), but in some designs it moves in a vertical plane, taking each horizontal layer of the stream in succession (Fig. 434). Ordinarily, it is operated by a water-balance or tilting box; occasionally, by independent motor (Fig. 435); and exceptionally, by the mill machinery (Fig. 436).

When, as with direct flotation and cyanidation, the crude ore is crushed fine before dressing or treatment begins, pulp samplers may be used to determine the assay value of crude ore. Their greatest application, however, is in the sampling of mill-products after treatment has begun, and in the sampling of residues flowing away after treatment is complete. Exceptionally, as when flotation concentrate is borne by water to dewatering appliances, pulp samplers are used to sample concentrate.

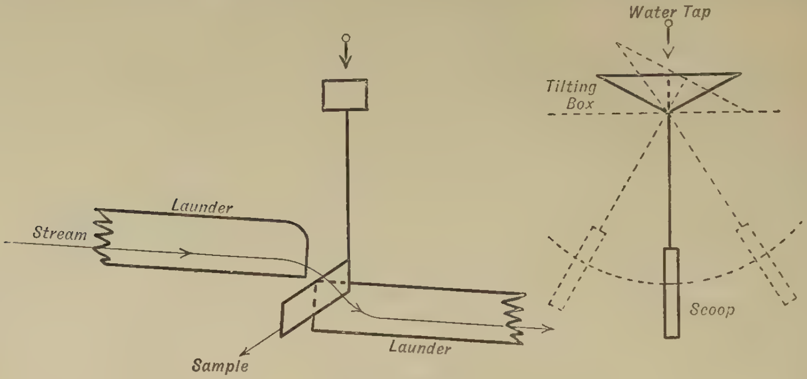

FIG. 433.

Pulp Sampler with Cutter moving across the Stream.—Diagram (p. 627).

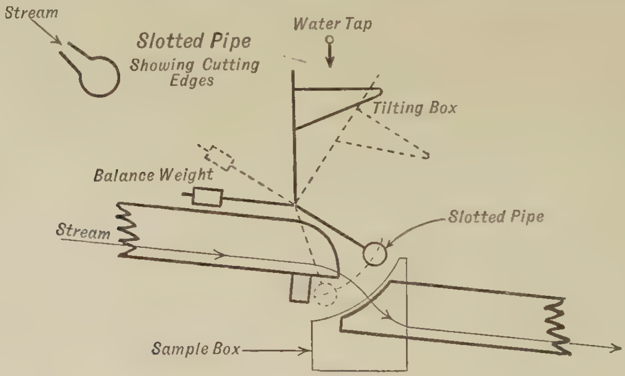

FIG. 434.

Pulp Sampler with Cutter moving down through the Stream.
Diagram (pp. 627, 630).

The sample accumulated over the given time is generally such as can be handled by laboratory means, that is to say, it does not usually exceed 100 lb., and in consequence need not pass in series through other similar samplers. The receptacle into which the cuts are directed is large enough to contain the sample accumulated per day or per shift, as the case may be. At the end of this period the supernatant clear water is decanted, and the sample carefully dried, after which it is divided by quartering or by riffling.

The proportion taken as sample is much the same as that represented by the ultimate sample delivered by dry samplers in series, and it is determined by similar considerations. On the one hand, it must be large enough to make the sample reasonably representative; on the other, it should not be too bulky. In general, rich and spotty ores or products are sampled in somewhat larger proportion, and poor and regular materials in smaller proportion. In practice, however, it will also be found that where the quantities passing are larger, the proportion sample is taken as smaller, and *vice versa*; with large quantities and low value a proportion of 1 : 5000–10,000 might well give a reliable result, though probably with ordinary quantities 1 : 500–1000, and perhaps still less, would be the proportion taken.

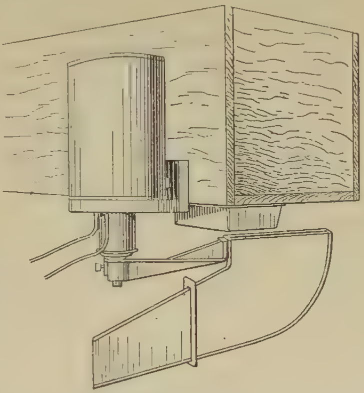

FIG. 435.

Flood Pulp-Sampler. — General View. The cutter of this sampler is actuated by two solenoids under control by a switch. A time-piece automatically operates this switch at desired intervals, closing the circuit for 1—5 seconds, the cutter being moved through 45° into a position to take the whole stream during that period; when the circuit is opened, the cutter returns to its starting point. One second at intervals of an hour is usual for regular material; for spotty material, half or quarter this interval (pp. 627, 630).

As with dry samplers, the proportion taken by any particular sampler is determined by the relation between the dimension of the cutter, the width of the stream, the speed of passage, and the number of cuts. The minimum width given to cutters may be taken to be a quarter of an inch and the maximum about one inch; the speed of passage varies from a measured movement by mechanism to an accelerated swing under gravity; and the number of cuts varies from 1 to 20 per hour. If the cutter take one-tenth of the stream at a time, the passage occupy one second, and the

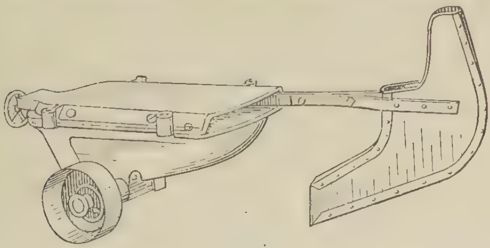

FIG. 436.

Scobey Pulp-Sampler.—General View. The cutter of this sampler is mechanically driven (p. 627).

number of cuts per hour be 10, then the proportion that the sample bears to the whole will be 1 : 3600.

By mechanism the passage may be so relatively slow and precise, and the errors of entry and exit from the stream so diminished, that the weight collected as sample may be accepted as a reliable index to the tonnage passing in a given time ; the pulp sampler in effect then becomes a time sampler. Where, however, the cutter is driven across the stream by the relatively instantaneous tipping of a tilting-box, no reliable index to tonnage is afforded (pp. 620, 634).

To permit the insertion of these pulp samplers the launder carrying the stream is broken, and dropped perhaps a foot. Considering the unconfined stream exposed at this break, it seems likely that the stratification set up in the launder will obtain also at the fall, that is to say, the mineral grains will be concentrated below, the gangue grains above, while the upper layers will contain the slime in suspension. With such a disposition it is likely that the cutter which moves horizontally across the stream will take a more reliable sample than one which moves down through it ; accordingly, the horizontally-moving cutter is the more common design. The sampler which moves down through the stream has, however, the advantage of traversing the smaller dimension of the stream, and consequently of taking a smaller sample ; a common sampler of this type is one having a slotted pipe as cutter, the slot facing upstream, and the pipe, while practically horizontal, yet sufficiently inclined to direct the sample to one side (Fig. 434).

The difficulty by reason of the stratification set up in the launder is sometimes contoured by dropping the stream through the bottom of the launder, the cut being taken as it falls (Fig. 435). Across so vertical a stream only a cutter moving in the horizontal plane is practicable.

Finally, it may be said that with the moving-cutter samplers a prerequisite to reliability is that the cutter shall move right across the stream, in at one side and out at the other, the stream continuing uninterrupted between the two extreme positions of the cutter. All other things being equal, it is conceivable that the cutter which, after passing through the stream, returns again to make a second cut while moving in the opposite direction, would give a more reliable sample than one which makes every cut in the same direction.

Reliability of Sampling.—Though there is a specific reliability of sampling, this reliability is not capable of expression except through the associated results of assaying, which themselves are only relatively precise ; accordingly, the reliability of sampling is here taken to include that of assaying.

Assaying may be taken to start from the sample, weighing about half a pound, delivered to the assayer. If this sample has been reduced to 80—120 mesh before delivery, then, generally speaking, by no procedure in mixing can a poorer or richer portion designedly be taken for the actual assay. Yet, by pure chance, such might very well happen with precious-metal ores, or with material not ground fine enough; a single mineral particle above or below the true average might then make the small portion taken for the actual assay appreciably richer or poorer, respectively, than the true average value. For this, and for the reason that separate assay determinations can never be conducted under precisely the same conditions, the reliability of assaying is not extraordinarily close; assaying is subject to both the errors of chance and those of manipulation. Speaking of crude ore, the assay determinations of silver ore have generally a probable error of 2 per cent, the result being generally higher than the true value; those of lead and zinc have much the same error; copper assays are somewhat more precise; gold assays, on the other hand, generally err more widely.

Like assaying, sampling also is subject to chance errors and constant errors. The errors of chance are those beyond absolution by care or skill; they are, however, impartial, and in a number of samples tend to annul one another. On the other hand, the constant errors, those due to faulty working of the sampling device or to irregular manipulation in the laboratory, remain.

Dealing first with the operation of chance, as indicated earlier, the probable error of the average of a number of observations—a sample may be regarded as an observation—is inversely proportional to the square root of the number. This relation, based upon the theories of probability, is contained in the Gauss formula which reads:

$$D = \pm 0{\cdot}6745 \frac{\Sigma(e)^2}{n(n-1)},$$

where D is the probable departure from truth of the average of n samples, n being a reasonably large number; and e, the probable departure of the individual sample.[1]

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[1] It may be recalled that in a number of separate observations conducted with equal care there will be a maximum departure or error in the individual observation, this maximum increasing as the square root of the number of observations; a mean error, which will be the simple average of the errors of the separate observations; and a probable error, which will be the error of the sample situated in a medial position with respect to its error. And that the mean of a number of observations is more precisely obtained from considerations of the mean squared departure from the mean observation than by simple arithmetical average, that is to say, more precisely obtained by the application of Poisson's formula,

$$F = \frac{\Sigma(e)^2}{n(n-1)},$$

where F is the mean error of the mean observation, e the departure of each observation from the mean, and n the number of observations.

From probability, therefore, the assurance is received that by averaging a reasonable number of samples, even though the probable error of the individual sample be relatively large, the chance error of the average can readily be brought within a reasonable figure. Assuming, for instance, that the mean departure from truth of each cut contributive to a daily sample be 25 per cent, and there be 120 cuts per day, then the probable error of the daily sample will be about 1·5 per cent, and that of the monthly average only a minor fraction of 1 per cent. It is clear, therefore, that the chance error in sampling is readily brought below the probable error of assaying; by taking samples as large as ordinary convenience permits it may even be considered in a practical sense as eliminated.

The constant errors in sampling allow no similar procedure in their elimination, but are only reduced by careful attention and checking. If a stream of ore has been properly sampled, the sample taken will represent the whole not only in value but in composition and mechanical character. The sample should, for instance, have the same sizing analysis as the whole, the same proportion of sulphides, etc. With pulp samples, the proportion of water to ore should likewise be the same as that known to obtain in the stream sampled. These are points which can be checked either by time samples or by bulk figures obtainable from the results over regular periods.. To secure this correct representation the cutter must pass through the stream smoothly, regularly, and without undue splash or scatter. A relatively large number of cuts is necessary; but an excessive number means the scattering of the larger pieces of a dry stream and of the suspended slime of a wet stream, these fractions not then being properly represented in the sample. Accordingly, there comes a point where increased speed means diminished reliability.

In sum, it may be said in respect to sampling, that though some ores are more difficult to sample than others, and though the error of sampling is additive to that of assaying, the assay value of the ore treated in a mill may be determined with reasonable accuracy, and usually within an error of about 2·5 per cent. Certainly, by an ordinary sampling-equipment accuracy in sampling *per se* is obtained as readily as, and perhaps more readily than, accuracy in assaying.

The necessary sampling-equipment, however, costs money both to purchase and erect, and to operate. For a mill of large capacity an adequate equipment might cost, say, two or three thousand pounds, while the operating cost would be of the order of a penny or two pence per ton of ore milled. Though these costs are not great, they must be viewed in relation to the benefits they bring; should these benefits be considered doubtful, or should the same control be obtainable more simply, to incur even such small costs would be unwarranted. In the milling of gold ores it is, for instance, widely held that the value of the ore entering the mill is better obtained by combining the output with the tailing assay. Admittedly, the output divided by the tonnage gives a figure independent of the hazards of sampling and those of the eventual assay of a very small portion; admittedly also, the tailing by reason of its low and regular value permits reliable determinations of its value in spite of those hazards. Yet the losses in the treatment of bullion or amalgam, as well as the losses by theft, are perhaps hardly as negligible as this composite method suggests; in any event, to ignore them is not only to run the danger of their increase, but to rob the control of the stimulus which an endeavour to obtain independently the real value of the ore would bring.

At custom mills and at smelteries the systematic sampling of the ore is accepted as being so rational and reliable as to be indispensable. It must be said, however, that the higher value of such ore permits the installation of a more complete sampling equipment and justifies a greater operating cost, this cost being generally about two shillings per ton, and sometimes as much as four shillings.

Weighing and Tonnage

The weight of the dry ore treated is equally as important as its assay value, the product of these two factors giving the total valuable content; the weight of the dressed products must also be obtained, that their total valuable content may be computed; that of the residue discarded is taken to be the difference between these two weights.

The dressed products, being relatively small in amount and high in value, are generally weighed directly on platform scales, after which a deduction is made for the moisture contained, this moisture being the subject of a special determination. Such direct weighing, though simple and accurate, is relatively costly, and therefore not generally applied to the crude ore, which is large in amount and relatively low in value. Moreover, reliable estimates of the tonnage treated may be made by fractional weighing, by continuous and automatic weighing, and even by volumetric measurement. These various methods are described below, beginning with those applied to roughly broken ore.

Truck Tally.—Knowing the number of trucks delivering ore during a given time, a fair estimate of the tonnage so delivered can be made by weighing a reasonable proportion of the truck-loads, say 1 in 20, daily. This method demands a suitable track-scale. The number of trucks may either be obtained by ordinary tally, which requires careful attention, or automatically by a counter. A suitable counter may be operated by a weighted lever raised by the loaded truck as it passes over a loose rail-end. Applied to ore which has been through the breakers, the calculated tonnage is accurate to within about 2 per cent, since broken ore fills evenly and regularly into the trucks and leaves them clean ; it is necessary, however, also to weigh a number of the returning empty trucks in order to get a reliable figure for the tare-weight. Applied to lump ore, or applied more crudely than described, the results will probably not be within 5 per cent of the truth.

Scale Car.—All the ore may be passed out of bins into a special scale-car, running along under the bin gates, each load being weighed before being tipped into bins below. This method is correct to within 1–2 per cent ; and the scale-car itself is cheaper than any automatic weighing machine. On the other hand, two bins and extra height are required.

Weighbridge.—Frequently, where the trucks in which the ore arrives are large, a self-registering weighbridge is used, the weigher only balancing the scale and turning a screw to punch the weight-card. Sometimes two or three trucks are weighed at a time. This method, like the scale-car, is correct to within 1—2 per cent.

Continuous Weightometer.—Where the ore arrives by belt-conveyor the weight may be taken continuously by the pressure of the stretch of loaded belt, generally about six feet, which is in the weighing machine. In the Merrick weightometer this pressure is balanced by a float partly suspended from the weighing beam and partly immersed in mercury (Fig. 437). By its end this beam operates a totalizing integrator operated at the same time by the travel of the conveyor, the product of these two factors being indicated as tons. These continuous weighing machines are usually correct to within one per cent ; they are cheap to operate, but rather costly to purchase.

Stream Measurements.—When a dry stream is regularly cut for sampling purposes, and the time taken by each cut is relatively long and can be calculated with reasonable accuracy, then the weight taken as sample may become a reliable index to the total tonnage. Used as a check, this application of time-sampling has given results in close agreement with direct weighing (p. 620).

As indicated earlier, however, it is particularly with pulp that time-sampling becomes serviceable in the estimation of tonnage (p. 630). The stream of pulp is turned for a given time into a receptacle where the amount which collects can be accurately measured. To obtain reliable tonnage estimates the given time should not be less than 20 seconds, when the maximum time-error with a stop watch reading to fifths of seconds would be one per cent, and the probable error about one-half per cent. After measurement the pulp is agitated that a proper sample to determine its specific gravity may be collected in a weighing flask; or a separate sample for this purpose may be taken from the stream. In the knowledge of the specific gravities of the pulp and the ore respectively, the weight of solids in the sample, and eventually that of the material treated in a given time, may readily be calculated. Where the quantities are small the weight of solids in the sample may be obtained after decanting and drying, by direct weighing.

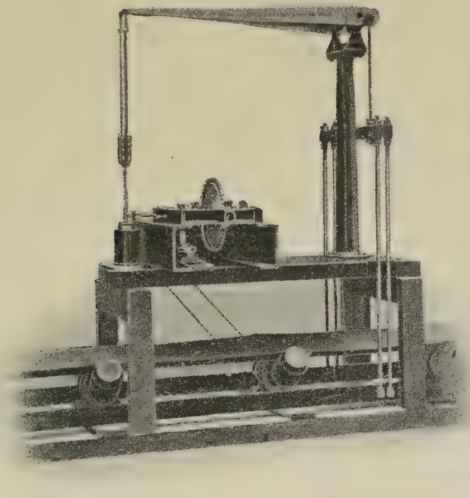

Fig. 437.

Conveyor-Weightometer.—General View of the Merrick Weightometer. The belt conveyor is seen in the lower portion of the illustration (p. 634).

Where the ore is regular in character and the supply is continuous, a determination of this sort undertaken daily will give a fair estimate of the tonnage treated. Where, however, the specific gravity of the ore is liable to vary, or the ore may vary from being soft to being hard, there is liable to be considerable error.

Tank Measurements.—Sand which has been borne by water into tanks and there collected is, after draining, in a condition permitting reliable estimates of the tonnage so collected to be made, when due care is taken. With the surface carefully levelled the volume of this sand is readily calculated. The other factor, the average weight of unit volume, is best determined by placing sheet boxes, perforated top and bottom, at various heights and places in the filling tank, these boxes being carefully recovered as the sand is eventually discharged, their contents being then dried and weighed. Several of these boxes are necessary because of local concentrations of the heavier and of the slimy portions. Such determinations should be made at regular intervals. Care must be taken that the results obtained are not applied to sand which has collected under other conditions; on the Witwatersrand gold-field, for instance, it was found that the sand collected under water weighed about 93 lb. per cubic foot; that the same mechanically transferred after drainage into another tank weighed only about 77 lb. per cubic foot; and finally, that after submergence under solution in the second tank the same sand increased its weight again to the intermediate figure of about 83 lb. per cubic foot.

The weight of slime collected in tanks, because of the impossibility of draining the remaining water, is not so readily determined. When brought into suspension, samples for specific gravity must be taken at various depths either by dipping them out or by tapping them through appropriate holes in the side. These determinations, even when made with reasonable care, are, however, not wholly reliable. Estimates of slime tonnage are best made when the slime is suspended in a stream, a condition which permits the specific gravity of the suspension readily to be determined; or when in the form of filter cakes of definite size and of dry weight determinable by weighing.

Reliability of Tonnage Estimates.—When the estimate of tonnage is made by direct or fractional weighing, the reliability of the estimate will still largely depend upon a careful determination of the moisture contained in the material. With crude ore such moisture determinations are best made upon material taken from the mill sampling-equipment; usually, after the stream in this equipment has reached the stage when the material has been crushed to about one-quarter of an inch, a weight of 5 lb. is taken, and carefully dried at about 110° C., after which it is weighed again. Even then an allowance has to be made for the moisture lost in the interval between the weighing for tonnage and the weighing of the moisture sample before drying; ordinarily this allowance will be an additional ten per cent to the moisture found, though the actual figure will vary with the climatic conditions and with the seasonal variations. The moisture in crude ore delivered to the mill usually varies from five to ten per cent.

When the estimates of tonnage are made from pulp measurements, the weights forming the basis of the calculations are always of dried material, and no further correction is necessary.

With proper correction for moisture, tonnage estimates of crude ore entering the mill, made intelligently, carefully, and impartially, are reliable; it is advisable, however, occasionally to check the method in force by another method. Instances are recorded where tonnages obtained by tank and slime measurements and also by stream measurements have agreed with those obtained at the weighbridge to within one-quarter per cent; similarly, the tonnage of flotation concentrate obtained by a weightometer has proved sufficiently accurate to provide a reliable check to direct weighing at the smelter. On the other hand, tonnage estimates made by officials anxious to report low costs per ton have sometimes been tragically in excess of the truth, while buyers of ore may sometimes consider that a ton should be as near a ton and a half as can be managed; accordingly, impartiality in those making the determination is a pre-requisite to reliability. Granted all the attentions indicated, the tonnage estimate should be within an error of one per cent.

With such reliability obtainable in sampling and tonnage respectively, by equipments and methods appropriate to the relatively low value of domestic crude ores, the total valuable content in the ore treated is computable within such a small error, say 2—5 per cent, that both interest and intelligence are stimulated, and a satisfactory control made available. The figure thereby computed will, it is true, not have the force of such absolute truth that, failing a complete rendering, a prosecution for theft or criminal negligence could be instituted, but it will be sufficiently precise to warrant that any discrepancy between it and the sum of the contents of the products should receive close attention from the staff; it will at the same time form a satisfactory basis in terms of which the valuable contents of the products can be expressed in percentages, a valuable means of control and comparison.

With custom ores and the closer sampling and weighing associated with such ores, the estimate of the total valuable content is reliable enough to form the basis of satisfactory purchase and sale.

RECOVERY AND ENRICHMENT

By dressing, the crude ore is separated into two products, a valuable concentrate and a worthless tailing, the former not being perfectly clean mineral, and the latter not perfectly clean gangue; the mineral then in the concentrate represents the "recovery," while that in the tailing represents the loss.

That these two factors may be on a basis for comparison with similar factors obtaining elsewhere, or with the work done from time to time at the same place, they must be stated in terms of the total valuable content of the original crude ore, either as fractions or as percentages of that content. In such terms the two factors are complementary, the one of the other, and the whole position with respect to them is declared by giving either. That being so, and recovery being the aim, recovery, either fractional or percentage, is the factor employed, the latter being usual. "Percentage recovery" is the number of valuable units recovered in the concentrate from every hundred units originally in the ore.

If the concentrate were completely of clean mineral or metal, the percentage recovery would be a complete and equitable index to the technical success of the operation. That is the position with gold and silver ores, where the particular treatment results in the recovery of bullion. It still remains a useful index when the concentrate is approximately clean, or when comparison is being made between operations producing concentrates of like though imperfect cleanliness. But it fails as a basis of comparison between the work of equipments producing widely-differing grades of concentrate, as also between that of machines in the same equipment, each advancing the work of dressing by stages not necessarily equal. Accordingly, recovery alone is not a complete index to the success of a dressing operation; it includes no element expressive of the cleanliness of the concentrate.

The factor commonly employed as the necessary corrective to recovery is the ratio between the weight of the original ore and that of the recovered concentrate, this being known as the "ratio of concentration." Reciprocally this ratio gives the proportion of concentrate recovered, and complementarily the proportion of gangue removed; that is to say, it gives indirectly an idea of the cleanliness of the concentrate. Perhaps a more direct expression of that cleanliness is afforded by the ratio of the assay-value of the concentrate to that of the ore, a ratio which may be described as the "ratio of enrichment." Neither of these ratios is, however, in a form which permits it to be directly incorporated with the recovery, to produce in a single expression a precise index to the whole success of an ore-dressing operation (p. 383).

Before indicating how such an expression may be constructed, the importance of the factors already described demands that they themselves and their relations to one another be defined with the greater clarity afforded by symbols and formulae.

Let a be the weight of ore treated, that is, of the feed, and x its assay-value;
b, the weight and y the assay-value of the concentrate;
c, the weight and z the assay-value of the tailing.

Then: ax is the valuable content of the feed; by that of the concentrate; cz that of the tailing; and $ax = by \times cz$.

Further: by, is the Recovery;

$\frac{by}{ax}$, the Fractional Recovery, R;

$100\,\frac{by}{ax}$, the Percentage Recovery, $100\,R$;

$ax - cz$, the Theoretical Recovery, or that called-for by assay;

$100\,\frac{ax - cz}{ax}$, the Theoretical Percentage Recovery.

$\frac{a}{b}$, the Ratio of Concentration, $R.C.$;

$\frac{y}{x}$, the Ratio of Enrichment, $R.E.$

All these expressions, except the last, are not determinable till the necessary weights are available. Yet the weights and assay-values of dressing products are interdependent to the extent that the more important of the above expressions may be given in terms of assay-values alone, in which form they usually are more quickly available; thus:[1]

$$a = b + c; \text{ and consequently } az = bz + cz.$$

Subtracting this last equation from that given above, the weight of the tailing is eliminated and the ratio between the two remaining weights is obtained in terms of the respective assays, thus:

$$ax = by + cz$$

$$az = bz + cz$$

$$a(x - z) = b(y - z)$$

$$\frac{a}{b} = \frac{y - z}{x - z}.$$

Continuing: $100\,\frac{by}{ax}$, the Percentage Recovery, becomes $100\,\frac{y(x - z)}{x(y - z)}$

$\frac{a}{b}$, the Ratio of Concentration, becomes $\frac{y - z}{x - z}$.

A special case is that of the precious-metal ores, with which, since the weights of feed, a, and tailing, c, are equal,

$100\,\frac{ax - cz}{ax}$, the Theoretical Percentage Recovery, becomes $100\,\frac{x - z}{x}$.

Were there no loss in dressing, the ratios of concentration and enrichment would be equal, the concentrate would increase in assay-value in the same proportion as it decreased in weight. Owing to loss, however, enrichment lags behind concentration, the relation between the two being the same as that between fractional and complete recovery. This relation appears from the expression for fractional recovery, $\frac{by}{ax}$, which, being regarded as $\frac{b}{a} \times \frac{y}{x}$, declares the fractional recovery to be the product of the reciprocal-ratio of concentration and the ratio of enrichment, and the ratio of enrichment to be equal to the product of fractional recovery and the concentration ratio. Thus:

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[1] Hoover, T. J., *Min. Mag.*, August 1910, p. 119.

$$\text{Fractional Recovery, } R = \frac{by}{ax} = \frac{b}{a} \times \frac{y}{x}$$

$$= \frac{1}{R.C.} \times \frac{R.E.}{1}$$

$$\therefore\ R.E. = R \times R.C.$$

Returning now to the formulation of an index properly combining recovery and cleanliness, the following considerations were put forward by Hancock.[1] Theoretically considered, the perfection of dressing would connote the complete removal of all the mineral from all the gangue; in other words, it would mean the complete recovery of all the mineral in a perfectly clean concentrate. That being so, the proportion of clean mineral recovered clean of all gangue, or, since one could not be achieved without the other being achieved to the same extent, the proportion of clean gangue removed completely from all mineral, would be the required index. Centering the considerations around the concentrate, a given concentrate must be considered as consisting of two materials, the one an amount of perfectly clean mineral representing the success of the operation, the other an amount of unaltered feed for which obviously no credit can be taken. The amount of perfectly clean mineral is that obtained by diminishing the total mineral in the concentrate by that amount which finds itself there as part of the unaltered feed; expressed fractionally with respect to the total mineral, or as a percentage of that total, that amount gives the required index. It will be found that this index is equally obtainable, fractionally, by subtracting unity from the sum of the fractional recovery and fractional gangue-removal; and as a percentage, by subtracting 100 from the sum of the percentage recovery and the percentage gangue-removal. Accordingly it combines the factors of recovery and enrichment. The calculation in percentage is as follows (see p. 292):

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[1] *Trans. I.M.M.*, Vol. XXVII., 1918, p. 111; *M. Mag.*, September 1918.

Let a be the weight of the feed, and x the percentage mineral content;
b the weight of concentrate, and y its percentage mineral content.
Then the feed consists of:

$$\text{mineral, } \frac{ax}{100},$$

$$\text{gangue, } \frac{a(100-x)}{100},$$

and the ratio of the weight of the feed to the weight of gangue in the feed is

$$\frac{100}{100-x}.$$

Further, the concentrate consists of:

$$\text{mineral, } \frac{by}{100},$$

$$\text{gangue, } \frac{b(100-y)}{100}.$$

Whence the percentage index becomes:

$$100\left\{\frac{\frac{by}{100}-\left\{\left(\frac{b(100-y)}{100}\right)^{\frac{100}{100-x}}\right\}^{x}}{\frac{ax}{100}}\right\}$$

$$=100\left\{\frac{by}{ax}-\frac{b(100-y)^{\frac{100}{100-x}}}{a}\right\}$$

$$=100\left\{\frac{by}{ax}-\frac{b(100-y)}{a\left(\frac{(100-x)}{100}\right)}\right\}$$

= Percentage recovery − percentage of gangue not removed.

= Percentage recovery − (100 − percentage removal).

= Percentage recovery + percentage removal − 100.

According to this index, perfect separation, represented by 100 per cent, demands a full recovery of the mineral and complete removal of the gangue. A full recovery with no removal, as would occur supposing the crushed ore were simply run into a pit, would have zero as its index. Zero would also be the index when perfect cleanliness were obtained only by incurring complete loss. Or, when in removing 50 per cent of the gangue only a 50 per cent recovery were made, this last case connoting a simple division of the material into two similar and equal portions.

When instead of two simple products, concentrate and tailing, a dressing plant or a dressing machine produces also a middling product, then, in computing the index, this middling product is treated in a second calculation as though it were a concentrate and the only one produced; if of higher value than the feed there will then be a positive figure to add to that resulting from the first calculation, while if of lower value there will be a figure to subtract.[1] Again, when there are two minerals present, each having its own recovery, the calculations will be facilitated by reducing the assay-values of the feed and products to money values, so that an average recovery, sometimes described as the 'economic recovery,' may be available.

Finally, in general, and assuming ordinary care, there is close agreement between the theoretical recoveries calculated from assays alone, and the actual recoveries calculated from weights and assays, this close agreement being particularly noticeable when, as in dressing plants generally, treatment is not prolonged (p. 664). Where, however, as in cyanide leaching plants, treatment may take many days, it frequently happens that the agreement from month to month is not close, the actual recovery may substantially exceed or be less than the theoretical. Even then it will generally be found that the divergence of one month will to a large extent be made good by an opposite divergence of the next month, and that from year to year the agreement is such as to give confidence in the work and in the control.

In addition to the possibilities of control offered by checking the actual recovery with the theoretical, and by comparing the recovery with recoveries at other mines or with those of other times at the same mine, useful control of the machines in operation is also possible by the employment of hand-testing machines, such as washing pans, plaques, etc., from which it may be seen at once, and without waiting for the assay, whether any machine is functioning properly. The men working in a dressing plant should be encouraged to make themselves expert in the use of such helpful means of observing more closely both the operation of the machines and the character of the ore being treated. Those who have had the opportunity of watching the Cornishman use the vanning shovel on the dressing floor will have admired the artistic skill he brings to this good purpose.

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[1] Hancock, *E. & M.J.*, April 10, 1920, p. 841.

CHAPTER XV

DRESSING SYSTEMS AND PLANTS ; APPLICATION AND RESULTS

Extent of Dressing

Considered commercially, the purpose of dressing crude ore is to produce a dressed ore, suitable either for the market directly or to be metallurgically treated for a marketable product. This purpose is only possible of being maintained when the monies received from eventual sale cover, in the first case, all the costs of mining and dressing, and, in the second case, all those of mining, dressing, and metallurgical treatment, while leaving as large a margin as possible for profit.

Dressing, therefore, is a middle operation, of which the raw material, crude ore, may be considered as purchased at the cost incurred in mining, and of which the finished article, dressed ore, is sold at the price it will fetch in the market or is worth to the metallurgist. As such, greatest profit, and neither perfect cleanliness nor complete recovery, is the over-riding consideration. Accordingly, the extent of dressing, whether in respect to enrichment or to recovery, varies considerably.

In the matter of enrichment the concentrate must at least be clean enough to be acceptable to the smelter. Some minerals require to be very clean ; tin concentrate, for instance, must assay at least 55—60 per cent of tin, though the ore may assay less than one per cent. With other minerals, the removal of gangue may stop long before it approaches completeness ; copper concentrate suitable for the smelter, for instance, often assays as low as 5—10 per cent.

Beyond this, cleanliness is only pursued so long as the better terms received for the concentrate more than make good the extra cost and loss involved in prolonging the dressing. Where the concentrate is shipped a long distance to smelters, the lower freight of clean concentrate would constitute a not unimportant item in such better terms.

Expressing enrichment by the ratio of enrichment, the higher the value of the particular metal of the ore being dressed the greater will this ratio be. It is immensely high with the precious-metal ores ; it is usually 30—60 with tin ores ; while with base-metal ores generally, it is 5—20. Again, with different ores of any one metal it is higher with poor ores than with rich ores; the ratios of enrichment with alluvial gold and tin ores, which are notoriously poor, are, for instance, of the order of 1,000,000 and 1000 respectively.

In the matter of percentage recovery, commonly spoken of as the "recovery," there is not the same wide range; if there is to be any likelihood of profit a substantial proportion of the valuable content must be recovered; but the great cost of recovering anything of the last few units precludes complete recovery. Within this range the most profitable recovery depends upon several factors.

It depends upon the richness or poorness of the ore; an extra proportion recovered from a rich ore might more than cover the extra cost, whereas the same additional proportion recovered from a poor ore might only result in loss. Rich ores allow more plant and a more extensive dressing, and accordingly high recoveries are associated with them rather than with poor ones. Poor ores often permit only the simplest dressing; a simple washing may suffice to remove the clay or sand from iron ores, a simple screening may remove the larger and poorer pieces from some poor gold ores, and leave an under-size rich enough for shipment or further treatment.

The recovery sanctioned in practice, also depends upon the degree of enrichment demanded or reached, high recovery being more readily achieved when the ratio of enrichment is low (p. 383).

Recovery depends again upon the complexity of the ore, both in respect to mineral composition and the state of aggregation; high recoveries are associated with simple and with coarsely-aggregated ores, rather than with complex ores or with those having the mineral grains finely interlocked. With a complex ore also, the more important mineral will be saved at the expense of any subordinate mineral, the recovery of which will be relatively low. Generally, no profitable recovery can be made by dressing from any portion of the ore which approaches the colloidal condition, and such material can usually with advantage be run to waste. The mineral in tailings discarded from a previous treatment is likewise difficult of recovery. Where the dressing of tailing dumps has been profitable it will usually be found that the profit was made on a relatively low recovery; 30 per cent was, for instance, the recovery on which substantial profit was made from tin-ore tailings in Cornwall. From the zinc-middlings dumped at Broken Hill much higher and practically normal recoveries were possible and made, because the zinc in them had not been impoverished by the previous treatment; moreover, the treatment applied was radically different to that from which these particular middlings resulted. Dump treatment by leaching, where that is possible, generally witnesses higher recoveries than those common to dump treatment by dressing.

Finally, recovery will depend upon the maturity and size of the under-taking. In the early life of a mine the most profitable recovery might well be that resulting from hand-picking, coarse concentration coming later, and complete concentration last. Similarly, with large undertakings different products may be separated for different and more suitable treatment, a differentiation which would not be warranted with smaller undertakings. Accordingly, high recoveries are associated rather with mature and with large undertakings. In the matter of plant size, poor ores have the advantage over rich ores that they generally occur in amount large enough to justify a larger size of dressing plant; they have a second compensating factor in that they are more regular in character and value, this regularity making for better recovery.

Clearly, being dependent upon so many factors and circumstances, no precise general figures of recovery can be given; the following figures, however, indicate the common range of recoveries for different metals on mature mines: gold, 80—95 per cent; silver, 70—90 per cent; tin, 60—80 per cent; copper, 65—90 per cent; zinc, 60—90 per cent; and lead, 75—95 per cent. In computing the recoveries of different metals from a complex ore only those portions of the metals regarded as valuable constituents of the respective concentrates are considered; any zinc in a lead concentrate would, for instance, not be recovered, and therefore could not be considered as adding anything to the value of that concentrate; lead in a zinc concentrate is in much the same position (p. 439).

The Dressing System and Flow-Sheet

The choice, grouping, and arrangement of the machines employed in dressing constitutes the "dressing system," the graphical representation of which is termed the "flow-sheet."

Mechanical dressing being of itself unintelligent and only capable of doing a set work, demands that the ore submitted to it shall be regular in value and quantity. Regularity in quantity is accomplished by mechanical feeders; regularity in value must be the care of the trained intelligence. This care may express itself in the mining of the different grades or kinds of ore in such relative quantities that a flow of average ore to the mill is maintained, this being the usual procedure; or the different grades may be separately treated in different mills or units, a system which, though not common, is practised on the Cobalt silver mines and on some Continental lead-zinc mines; or the different grades may be separately delivered to the mill, where rich and poor campaigns are run in succession, this procedure being quite exceptional.

In the ordinary procedure, namely, that which aims at delivering average ore to the mill, regularity in the ore to be mechanically dressed is the more surely assured if previous hand-picking be part of the complete system. By such hand-picking rich mineral is removed, or worthless waste, or both, with the result that the remaining mill-ore is more regular both in value and character.

The choice of machine for the dressing of any particular ore will depend largely upon the properties of the ore itself, a matter discussed when describing the machines and processes. Their grouping and arrangement remain to be described. Practice in this respect may be divided under the following descriptive if somewhat artificial headings :—

Stage Concentration.—Generally, ores contain coarse as well as fine mineral ; often, therefore, at a relatively coarse stage in the size-reduction, coarse mineral grains are released in sufficient amount to warrant their separation by a concentrating appliance before the next stage in crushing is undertaken (pp. 177, 378, 471). The repeated separation of a certain amount of clean concentrate after each successive size-reduction constitutes stage concentration. Stage concentration is never practised narrowly, nor alone, but in conjunction with 'class concentration,' the broad stages of coarse, fine, and slime concentration, are often well marked (Figs. 208, 328). Seeing that tailing may likewise be discarded at each stage, to the extent that stage concentration is employed, the amount of material sent forward to be reground is diminished, and the loss associated with the wasteful crushing of already-released mineral is avoided ; moreover, by reason of its better physical condition, coarse concentrate is always more readily marketed than fine concentrate (Fig. 335). Further, stage concentration favours the application of particular processes and machines to material for which they are particularly suited ; coarse sulphides are well and satisfactorily recovered by water concentration, while the recovery of fine sulphides may be fitly left to flotation ; at the Anaconda copper mine, for instance, water concentration recovers everything above 2 mm., leaving the reground material and the original slime to be treated by flotation. Concentration should therefore begin at that stage in crushing when, all things considered, an adequate release first obtains.

Class Concentration.—Where the ore is so aggregated that reasonably complete release is obtained by relatively coarse crushing, stage concentration will then be associated with 'class concentration,' wherein different sizes of the crushed product are concentrated separately, a clean concentrate and clean tailing being made by each machine, and, in the coarse stages, perhaps also a middling product (Figs. 206, 282, 328). Stage crushing is then limited to the further reduction of the middling products.

Roughing Concentration.—Where the ore is poor and it would be difficult or wasteful to make clean concentrate at once, the initial operation may be directed to the removal of clean tailing, in which direction, of course, the effort is equally well expended. Since only a rough concentrate is then produced, such an operation is described as 'roughing' or 'ragging' (p. 379). In turn this rough concentrate is submitted to a second operation, in which it is cleaned, and which, accordingly, is described as a 'cleaning' operation (Figs. 226, 227, 285, 307, 330, etc.).

Roughing concentration, generally speaking, is not associated with stage-crushing, the material either not warranting such a refinement, or being already in a fine condition. Nor is roughing concentration associated with the separate treatment of different classes, all the material is treated together. Where applied to the treatment of crushed ore yet untreated, it permits, by reason of the relatively rough work first done, a high capacity to the concentrating machines, whether jigs, tables, or flotation machines. The water-concentrating machines adapted to this use, work with a depth of bed upon them; roughing tables, for instance, have riffles which extend right across to hold a bed behind each, to the exclusion of ordinary cleaning space. Having such beds the roughing machines are capable of absorbing variations in the quality and quantity of the feed, while passing a regular feed to the more sensitive and delicate cleaning operation.

Though long practised in Cornwall in the treatment of tin-ore slime (Fig. 256), its application to the whole of the crushed ore first became settled practice at Joplin on zinc ores, spreading thence to the lead ores of South-Eastern Missouri; finally, its benefits being realized, it supplanted preliminary sizing and classification in the concentration of low-grade copper ores (Figs. 255, 440).

Roughing concentration may be regarded as concentration by gradual enrichment; ordinary concentration, on the other hand, producing a small amount of clean concentrate at once, may be regarded as connoting rapid enrichment (p. 382). In the inclined-table treatment of slime in Cornwall gradual enrichment is closely followed, the concentrate made at each operation being only about twice or three times the value of the feed (p. 333). This slow enrichment involves a considerable extent of bed to hold the relatively great bulk of the collected concentrate (Figs. 256, 281); were more rapid enrichment adopted a smaller bed would suffice, and by retreatment of the tailing the same result would be obtained from a considerably smaller extent of total surface.

Retreatment Concentration.—This variation in the concentration system, which in its simplest sense connotes the retreatment of the tailing after the removal of a concentrate and without further crushing, is practised where the material is poor, and the recovery of the mineral largely depends upon repeating the operation. The many cells of the Joplin jig, those of flotation machines, constitute retreatment series. Retreatment of an impoverished tailing is also practised on slime tables, blanket strakes, etc.

It remains to be said that though one or other of these different systems of concentration may be a feature of a particular flow-sheet, there is no flow-sheet but what finds all these systems represented.

Further, that though particular crushing and concentrating machines do their best work on particular products, the number of steps and the number of machine-types should both be kept low, the complication of great differentiation should be avoided. Differentiation in treatment should wait till the assembling quantities ensure complete and continuous work for standard machines. The more complex the ore the greater the justifiable differentiation.

Milling.—As indicated at the outset, dressing, the mechanical preparation of the ore, may neither involve enrichment nor recovery, but simply the reduction to a fine or powdery condition. Such dressing may be described as 'milling'; it is the more usual preparation for metallurgical leaching.

The extent to which milling is carried will depend upon the character of the ore; when the ore is porous and the ore-mineral readily soluble in the extracting solutions, a reduction to $\frac{3}{8}''$ may suffice, as at Chuquicamata, Chili, where the ore-mineral is chiefly the basic sulphate of copper, brochantite, and in Arizona, where it is chiefly malachite; on the other hand, where the ore is compact and impenetrable and the mineral finely distributed, a reduction to slime may be necessary, as in the treatment of many gold and silver ores.

Where the necessary comminution is extensive, milling becomes so important an item that the term is given a wide sense, and purely metallurgical operations are included under it. It is, for instance, common practice to include amalgamation and sometimes even cyanidation, under milling, though more precisely they should be separately specified.

The 'flow-sheet' depicts so well the treatment of a given ore that this term is commonly taken to stand for the 'dressing system,' this latter term being uncommon. In its precise sense as a picture, various styles of flow-sheet are seen. The simplest is just a terse textual description of the system, ordered around arrowed directions of the flow, and there is no attempt to represent the different machines pictorially (Fig. 438). From that, the introduction of simple outline drawings of the machines is no great step (Fig. 439). The next step sees more elaborate pictures of the machines, the flow of the water being indicated in addition to that of the ore (Figs. 319, 440). Finally, the most elaborate flow-sheets give the values and quantities of the different streams, that is to say, they present the details of the whole "circulating system" (Figs. 161, 256).

In such flow-sheets it does not appear necessary to represent separately all the machines working in parallel, that is to say, doing the same work, though that is often done, the flow-sheet then becoming a key-plan to the whole installation. Seeing that flow-sheets serve primarily to expose clearly the principles of the particular dressing system, the machines in parallel appear sufficiently represented by one of their number, the actual number being indicated by a numerical figure (Fig. 332).

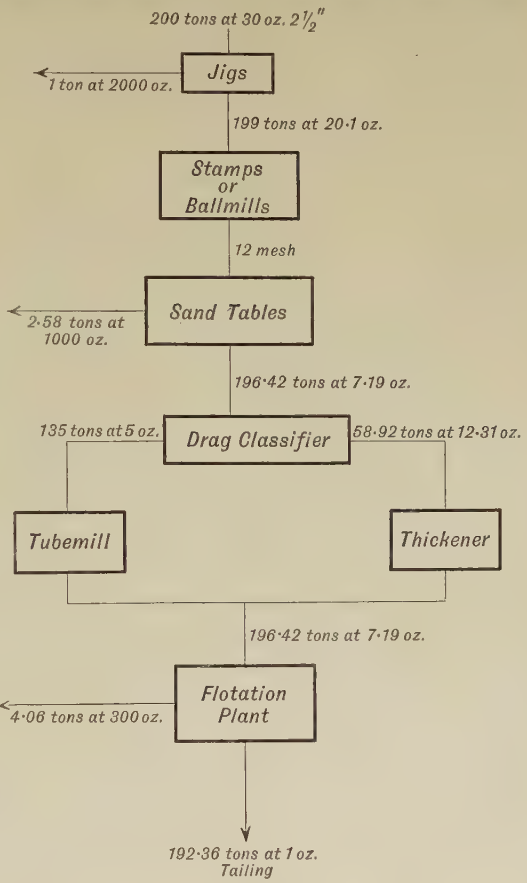

Fig. 438.

Water-concentration and Flotation at Cobalt, Ontario.—Flow-sheet. (Callow and Thornhill, *Trans. Can. I.M.E.*, Vol. XX., 1917, p. 28.) Two grades of ore occur, the one, high-grade vein-rock consisting of native silver, argentite, proustite, pyrargyrite, with niccolite, smaltite, etc., in a gangue of calcite; the other, low-grade wall-rock consisting largely of conglomerate and assaying about 25 oz. per ton. These grades are mined separately. Crushing the low-grade ore to 20 mesh, sufficient silver is released to permit a tailing of 3·5 oz. The better recoveries now being made are associated with finer grinding (pp. 648, 660).

THE DRESSING PLANT

The particular flow-sheet adopted for a given ore is determined by the character of that ore and the extent of the body it is desired to beneficiate, decision being made after appropriate tests, and full consideration of experience elsewhere. Though, rightly, the ore actually exposed will thus receive every consideration, some consideration must also be given to the chance of possible change in the future. With steep-standing deposits, for instance, the particular depth-zone in which the exposures find themselves, whether the oxidation zone, the enrichment zone, or the primary zone, should be recognized. Depth generally brings a change from the oxidized to the sulphide condition, it brings often also a change from coarse grain to fine grain, and occasionally a change from one metal to another, from copper to tin as in Cornwall, from lead to zinc as in many places.

The proper size of the plant will depend upon the amount of ore exposed and upon the carefully weighed probabilities of the future. Within limits a large plant is more economical in operation than a small plant, the fixed and overhead charges being divided over a greater tonnage. This limit may be put at a capacity of about 600—1000 tons per day, beyond which, greater capacity is best provided by adding other independent units.

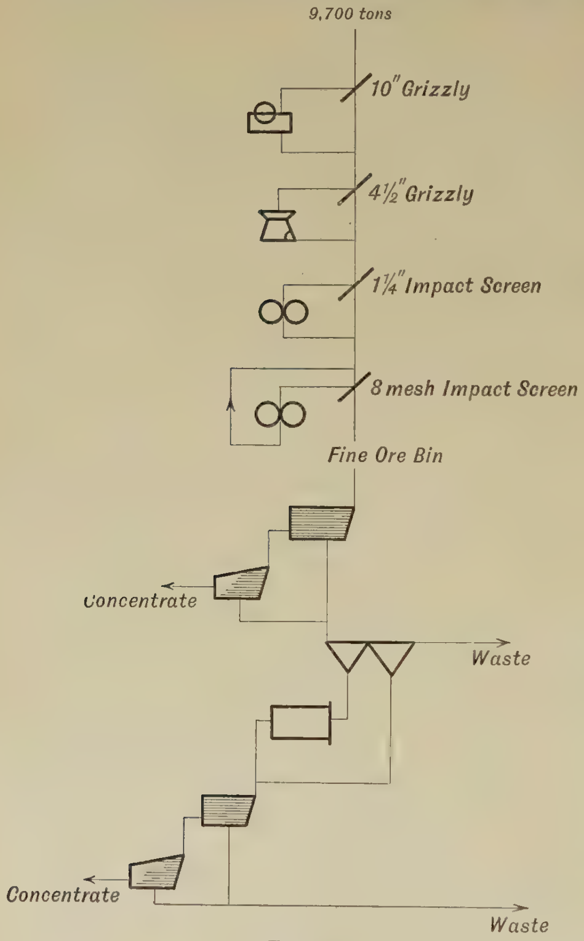

Fig. 439.

Water-concentration at the Alaska Gold Mines.—Flow-sheet. The capacity of this mill, completed in 1915, was 9700 tons per day. A little sorting is done during breaking; amalgamation was relinquished. The concentrate produced is, after some regrinding, retreated by tables, a galena concentrate with some free gold being marketed while the pyrite is washed away (pp. 649, 660). (*E. & M.J.*, May 7, 1921, p. 632.)

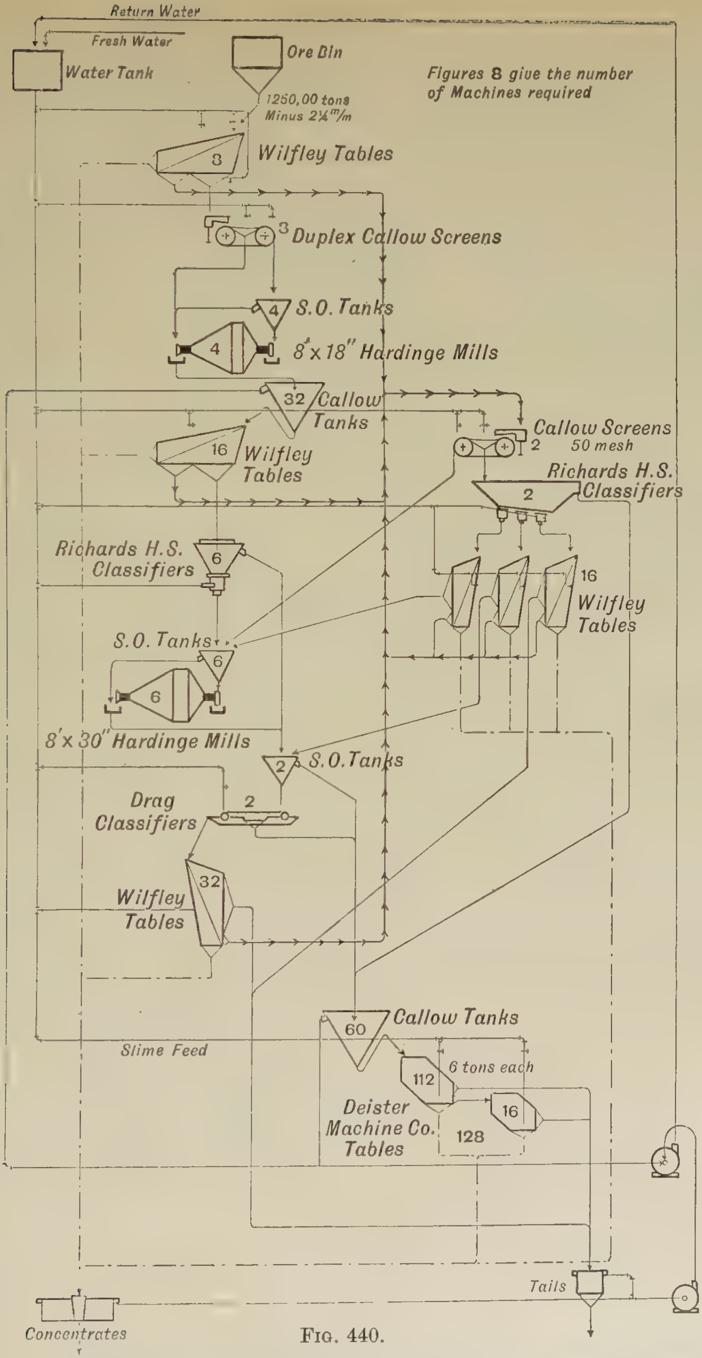

FIG. 440.

Water-concentration at the Porphyry-coppers.—Flow-sheet representing a scheme of water concentration projected, but abandoned in favour of flotation. The chief point about this flow-sheet was the gradual impoverishment of the slime by repeated treatment on roughing tables ; with classifiers in the place of roughing tables the assay-value of the material separated as slime would have been twice as high. As much as 75 per cent of the mineral finer than 200 mesh would be obtained from the roughing tables (pp. 647, 649). (Callow, *Trans. A.I.M.E.*, Vol. LVI., 1917, p. 678.)

Limited by the necessity to work smoothly, the capacity of a plant should be pushed till the greater economy of the larger quantity is offset by the additional loss in the tailing.

In respect to position, the dressing plant should be as near the mine as possible, in order to come under the same general supervision and to enjoy the same general services and amenities. If away from the mine it should be along the line of communications, and preferably downhill. In such a downhill position not only would the transport of the ore to the mill be favoured, but the necessary water would probably be more readily available. Most important also, the position should include a good mill-site and enough space, including dumping space, for present and probable requirements.

Where possible, a sloping mill-site is preferred (Fig. 441). Height is required by all dressing plants in order that the material may by gravity flow through the machines ; where there are many stages the necessary height may be as much as 120 feet. A site having a slope of about 20° will provide this height within a convenient distance down the slope, and permit terraces of convenient width without unduly high retaining-walls ; the pulp will flow by gravity, quietly and continuously, not only through the machines, but from machine to machine in succession, without the necessity of any elevating machinery ; each portion of the plant and practically each piece of machinery will be on solid foundations, and on the uppermost terrace the breaker may take its place without racking the building or structure (Fig. 442). These solid advantages are bought at the cost of the necessary excavations and retaining-walls, and, since a sloping site means climbing, lifting, and lowering, at the price of a little inconvenience in attendance and supervision.

On a flat site it is usual to have a flat mill, the material being elevated between stages, or between every two stages (Fig. 443). Such a mill is cheap to construct as there is little structure or excavation, and the elevating machinery which then becomes necessary is not costly. Elevators, however, incur considerable expense in maintenance, and are liable to bring irregularity and disturbance into the flow ; moreover, continual elevation and re-elevation causes loss of mineral by attrition. On the other hand, supervision and attendance are facilitated.

If, however, the breaking plant be kept separate, which on account of dust and vibration is generally desirable, and if the number of other stages be not great, a sloping mill may by appropriate structure be constructed even on a flat site. The requisite elevation is then done once and for all at the head of the mill, either by a vertical elevator, an inclined tramway, or better still by an inclined conveyor (Figs. 444, 445, 446).

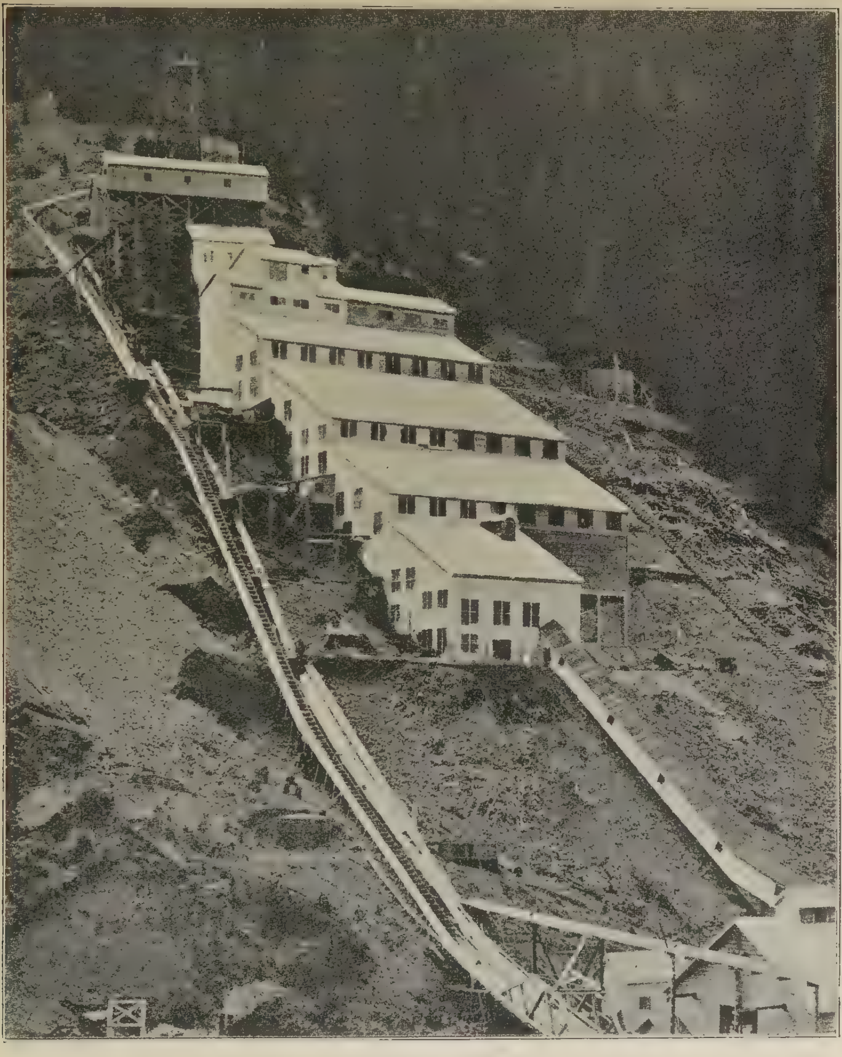

Fig. 441.

Sloping Mill-site.—General View of the Engels Mill, California. This is a flotation mill; the several terraces stand out clearly, as also do the inclined service tracks on either side (p. 652).

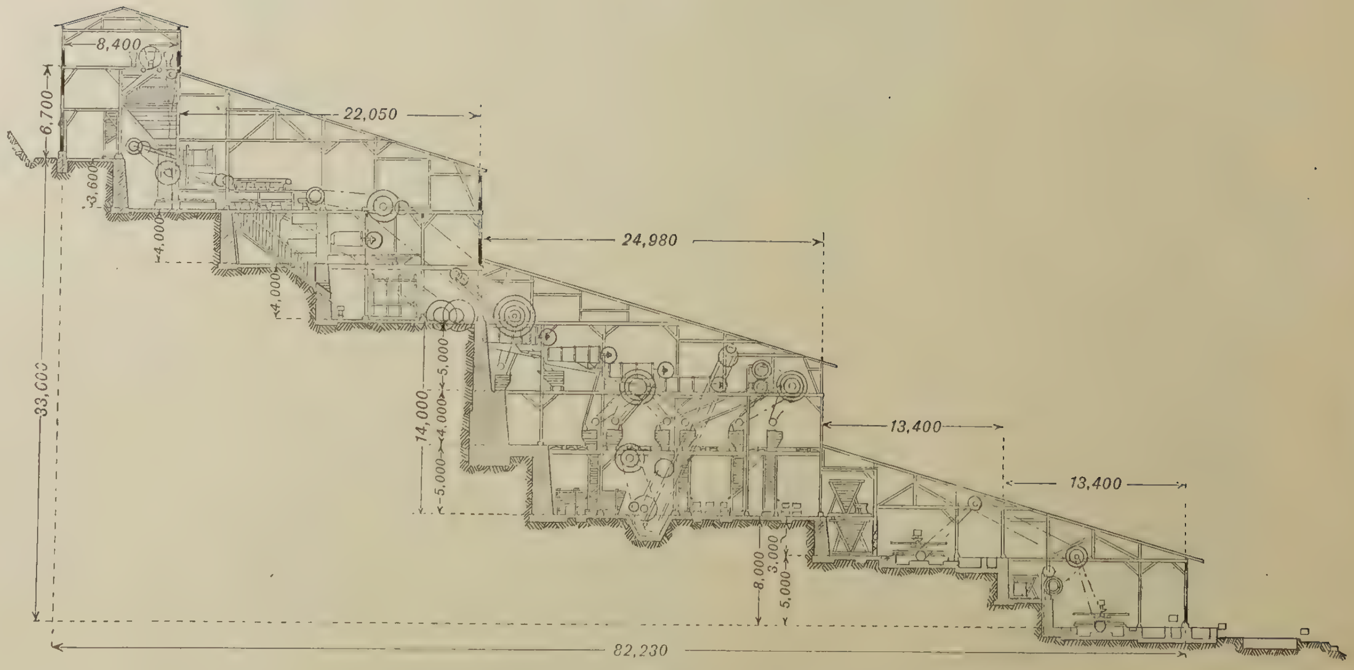

FIG. 442.

Sloping Mill-site.—Sectional view of the water-concentrating lead-zinc mill at Lüderich, Germany. Washing trommel, sorting table, rolls, sizing trommels, jigs, classifiers, and round tables, are all seen in sequence (p. 652).

Keeping the breaking plant separate it may be placed at the mine, when the additional advantage accrues that broken ore and not lump ore is transported; such broken ore is less destructive to trucks, fills them more fully and regularly, and permits more reliable estimates of tonnage. In addition to a breaker station at the mine, there is sometimes a fine-breaker station at the mill where dry-crushing is carried further; this second station likewise would be kept separate from the concentrating machinery.

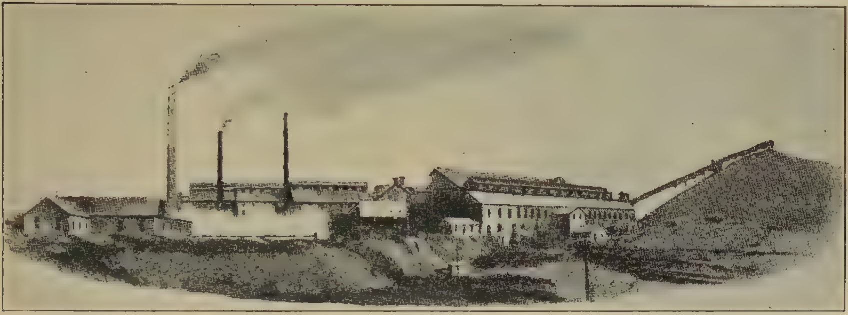

FIG. 443.

Flat Mill-site showing Tailing Stack.—General View (p. 652).

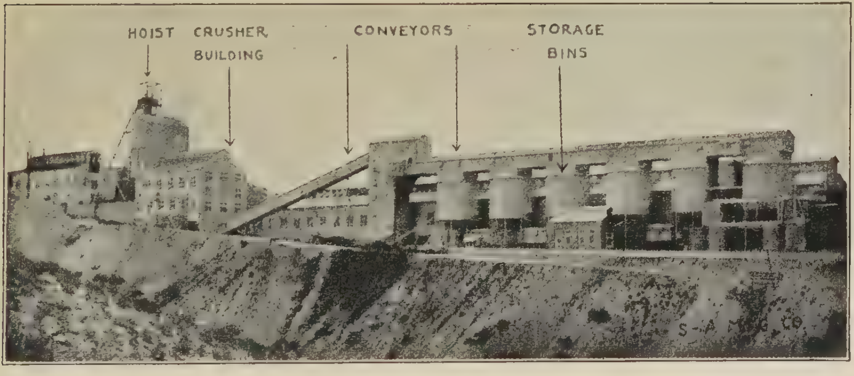

FIG. 444.

Flat Mill-site.—General View of the Inspiration Mill, Arizona. The actual mill is seen behind a row of storage bins; it is separate from the breaker station, connection with which is by an inclined conveyor (pp. 652, 653).

In determining the number of machines for a given capacity the details of the circulating system should be carefully calculated, the amounts appearing from these calculations being regarded as minima. Fluctuations in the system will arise by uneven distribution of coarse and fines in the bins, by idle circuits when screens and jig-sieves become blinded, or when rolls become inefficient; to meet these fluctuations machines should normally be run light. In addition, machines such as elevators, which while subject to great wear have yet the continuity of operations dependent upon them, should be in duplicate. Continuity of operation is further secured by placing above the breaker station and at the head of the mill, bins of sufficient capacity to tide over possible temporary disturbances (Figs. 46, 48, 383). With those attentions paid, many mills run fully 90—95 per cent of the full-possible running-time.

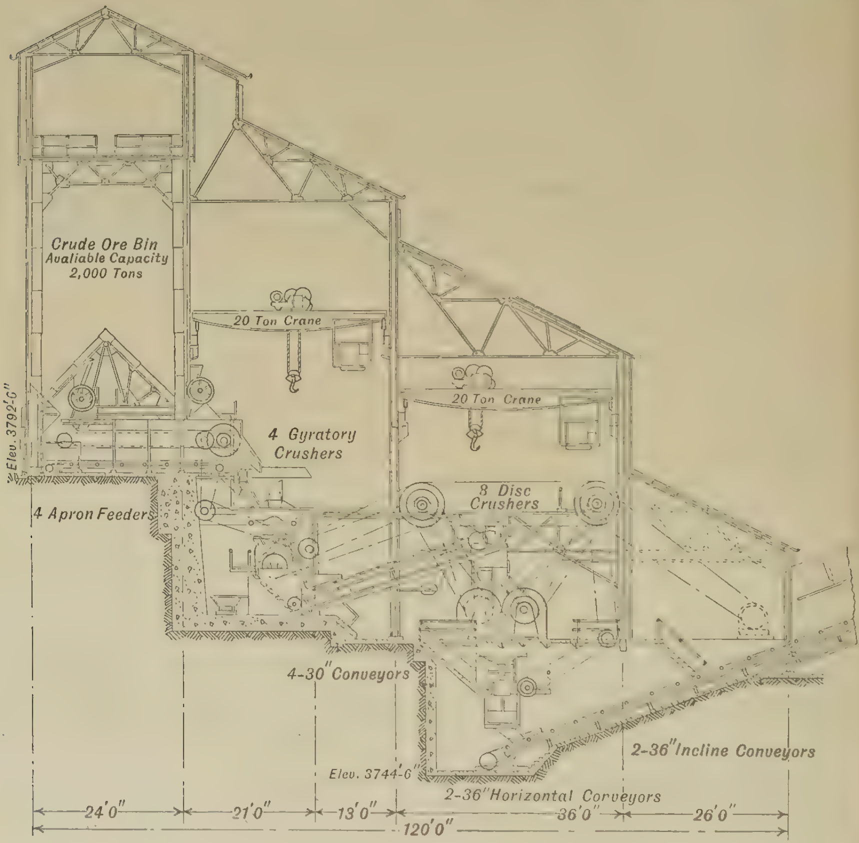

FIG. 445.

Breaker Station at the Inspiration, Arizona.—Transverse Section (p. 653).
(Burch, *Trans. A.I.M.E.*, Vol. LV., 1917, p. 707.)

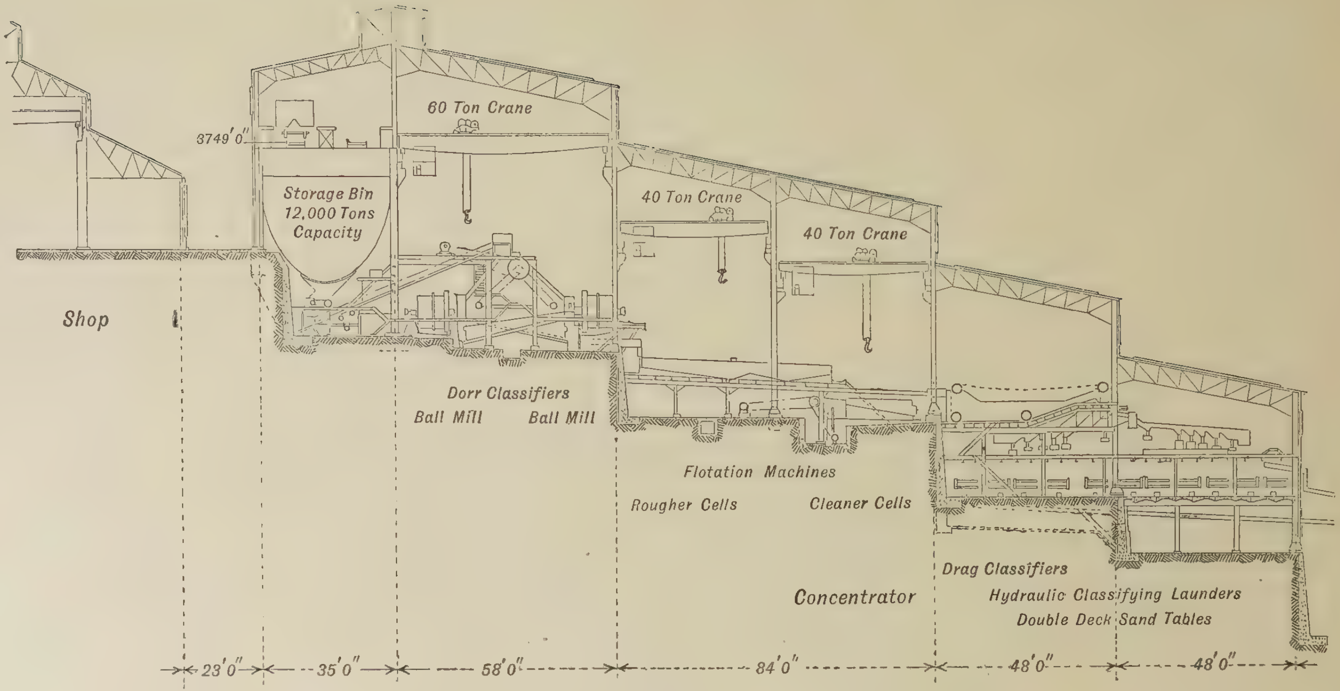

FIG. 446.

Mill at the Inspiration, Arizona.—Transverse Section (p. 653). (Burch, *Trans. A.I.M.E.*, Vol. LV., 1917, p. 707.)

The cost of appropriate mills varies with the extent and difficulty of dressing. Simple mills, those at Joplin for instance, cost £15—£20 per ton of daily capacity; water-concentrating mills, fully equipped to make an adequate recovery from difficult ores, will cost about £100—£150 per ton of daily capacity; flotation mills, £50—£100 per ton; magnetic mills, from £50 per ton for magnetite to £150 per ton for franklinite; and finally £150—£200 per ton for complete cyanide mills, all these figures including an allowance for the necessary power plant. Where the total recovery is made partly by one process and partly by another, partly by water and partly by flotation, partly by cyanidation and partly by concentration, the total cost of the mill will be somewhat higher than where a single process suffices. Where roasting is part of the treatment, the roasting plant itself will cost £120—£160 per ton of material roasted per day; similarly, a drying plant will cost about £20 per ton dried per day. The respective factory-costs of all these plants will generally be about one-half the costs stated, transport and erection being responsible for the other half.

The power consumed in dressing varies equally widely, namely, from about 5 k.w. hour to about 25 k.w. hour per ton; and the power installed from about 0·3 h.p. to about 1·5 h.p. per ton of daily capacity. Of this power consumption 60—90 per cent is generally required for comminution, while, itself, it represents about 30—60 per cent of the total power consumed on the mine.

Allowing that the water passing through the plant is stored and returned to service, and that during service and storage 25 per cent is lost by evaporation and leakage, the amount of fresh water required would generally be about 2—3 tons per ton of ore treated.

APPLICATION AND RESULTS

Crude ores are beneficiated by various processes, some dressing, some metallurgical. The more important of these, and the extents to which they are respectively applied, appear from the following statement, the figures of which are an estimate of the position some five or six years ago:—

Flotation Concentration	30	million tons per year.
Copper Smelting	30	,, ,,
Water Concentration	25	,, ,,
Gold-silver Milling	12·5	,, ,,
Magnetic Concentration	3·0	,, ,,
Lead Smelting	2·5	,, ,,
Copper Leaching	2·0	,, ,,

This position, since altered to the advantage of flotation and possibly also to that of leaching, illustrates the competition which dressing meets from direct metallurgical treatment. Many mines, and particularly copper mines, produce some ore which can be directly smelted. In the south-west copper region of the United States (Arizona, New Mexico, and Sonora) 5 per cent copper ore is smelted direct and with good profit, and exceptionally ore with even as little as 2·5 per cent; such low-grade smelting ores usually come from contact-deposits associated with limestone. On the other hand, with the disseminated copper deposits of the same region, these being associated with monzonite, porphyry, schist, etc., dressing is all-important, the ore generally assaying less than 2 per cent.

Leaching similarly challenges dressing with its competition. The ore of the immense deposit at Chuquicamata, Chili, assaying about 1·75 per cent of copper, is roughly crushed and then leached, dressing taking little part in its beneficiation. The sand tailing, piled up in former years from the Lake Superior copper mines, containing less than 0·5 per cent of copper, is successfully leached, dressing in the form of flotation having only established its right to treat some of the current slime-tailing.

Leaching and concentration may even partake in the beneficiation of the same ore. Some oxidized copper ores at Anaconda and Utah, for instance, are crushed to $\frac{1}{4}''$ and then leached with sulphuric acid, the residue from this treatment being reground and floated. At the Arizona Copper, on the other hand, 30 per cent of a 3 per cent copper ore is first recovered by an oxide-concentration plant in a form acceptable to the smelter, the residue being leached with sulphuric acid; a similar sequence is followed at Collahuasi, Chili.

Concentration and leaching likewise frequently co-operate in the beneficiation of gold and silver ores. The advantage may be that, with the relatively coarse and intractable mineral removed, the strength of the cyanide solution and the time of contact between ore and solution may be substantially reduced; or, the concentrate recovered may be of value for the base metals, lead, copper, etc., it contains; or, by concentration, minerals capable of actively interfering with cyanidation, soluble copper minerals, stibnite, etc., may be removed. Concentration by water is generally that employed in this particular association of concentration and leaching.

Concentration *versus* straight cyanidation is a question settled only on the merits of each case. If the heavy minerals contain no cyanicides and are not present in considerable proportion, it will probably be better not to concentrate. But if they are abundant, and a substantial portion of the precious metal is associated with them, they will probably deserve the special treatment only possible when in the form of concentrate. On the Witwatersrand, with about 3 per cent of auriferous pyrite, cyanidation without concentration is practised. At Porcupine, Canada, it is held that when the auriferous pyrite amounts to more than 4 per cent, concentration would probably be advantageous, and at the Hollinger about 15 per cent of the recovery is obtained from concentrate. The concentration of auriferous pyrite is also the established practice on the Mother Lode, California, where indeed it first began; after amalgamation in mortar boxes and on plates the tailing is concentrated on vanners or canvas tables, after which it may be, but generally is not, cyanided. On the large though low-grade Alaskan mines there is neither amalgamation nor cyanidation of the crude ore, but simple concentration of the auriferous sulphides (Fig. 439).

Where cyanidation is applicable, flotation has little chance of successfully competing with it in the treatment of the fine material, largely because cyanidation produces bullion, while flotation produces a concentrate not so readily marketed. Flotation would, however, appear to be a possible alternative to cyanidation in the treatment of such siliceous silver ores as contain the silver largely in the form of granular argentite, though no such displacement of cyanidation is recorded.

On the Cobalt silver-field, Ontario, the battle between water concentration, flotation concentration, and cyanidation, favoured first one then the other. In 1916 the milling-ore on that field was treated in the following different ways on different mines: water concentration throughout; water concentration for coarse material and sand, flotation for slime; water concentration for coarse material and sand, cyanidation for slime; coarse water-concentration, cyanidation of sand and slime; flotation of cyanide tailing.[1] At a later date flotation had increased at the expense of water concentration, but not at the expense of cyanidation; though by flotation it was possible from tailing assaying 6 oz. of silver to recover 80 per cent in a concentrate assaying 300 oz., there were difficulties in marketing such low-grade and, from a smelter's point of view, undesirably fine material; cyanidation which produces bullion accordingly prevailed for the treatment of the fine material and slime; in one instance, the Nipissing, there was a complete return to water concentration and cyanidation. Later still, beneficiation was achieved in the following different ways: water concentration and cyanidation; flotation and cyanidation; water concentration and flotation (Fig. 438).

A further instance of the application of both leaching and concentration in succession to the same ore, is afforded in the beneficiation of the tin-silver ores of Bolivia, these ores being first leached for the recovery of the silver and then concentrated for the tin. At the Sovocan mine, for instance, after crushing in dry ball-mills to 1 mm., the ore is roasted with 4 per cent of salt, then leached with water to remove copper, and with sodium hyposulphite to recover the silver, and finally concentrated for the tin.

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[1] Callow and Thornhill, *Trans. Can. M.I.*, March 1917.

The variety of beneficiation processes, and at the same time the progress in the art, is well illustrated by the different treatments successively applied with passage of time at the same mine, or on the same field.

At Broken Hill, New South Wales, for instance, the early days saw the direct smelting of the lead carbonate found below the gossan. Leaching and amalgamation were next applied, the former to recover silver chloride, and the latter native silver from siliceous ores associated with kaolin. Then, in the sulphide zone, lead was recovered by water concentration, and with it much of the silver; while, to-day, beneficiation is complete with the application of flotation to the recovery of the zinc.

At Mount Morgan, Queensland, the auriferous gossan was worked in 1886 by stamps and amalgamation, though without much success, and later by barrel chlorination, with a much better recovery. Then, the ore becoming more siliceous and sulphidic, roasting became necessary in preparation for chlorination. Next, in 1905, with increased sulphide copper-smelting began; eventually in 1912 chlorination was given up, all the ore being smelted. To-day, that part of the ore which is too siliceous for direct smelting is concentrated by water and flotation.

At Braden, Chili, beneficiation first began with hand-picking and reverberatory-smelting for matte; then water concentration similar to that employed on the low-grade disseminated deposits of the United States was adopted. Now, coarse water-concentration and fine flotation make concentrates which are nodulized and then smelted in blast furnaces.

RESULTS

For purposes of comparison, the results obtained in the dressing of any particular ore are sufficiently contained in a statement giving the tonnage treated during a given period, generally one month; the recovery and the ratio of enrichment obtaining over the period; and the cost of the operation. Of these factors the recovery is given as percentage recovery, while the ratio of enrichment is obtainable from the declared values of feed and concentrate, respectively.

The cost generally given is that incurred per ton treated, the actual figure being obtained by dividing the total expense for labour, fuel, stores, and supervision, during the given time, by the tonnage treated; expense necessary to maintain the plant in an efficient condition is usually included. Provided recovery and enrichment are satisfactory, this figure, generally described as the "operating" or "working cost," is a fair index to the care and activity of the working staff.

Since, however, the more complete the mechanical equipment the greater the tonnage a plant will treat for the same total expense, plants can only be fully compared with one another when to the working cost an amount is added to represent the cost of the money expended in the purchase and erection of the plant. The cost of money being the interest it would earn if invested, this additional cost, described as the "investment cost," is obtained by dividing the interest the money could be considered as capable of earning in a given time, by the tonnage treated in that time; ordinarily, while neither given nor derivable from the regular statements of results, it is about one-tenth of the working cost.

Finally, the loss sustained in the operation should in any precise comparison be entered as a cost against the operation. Though not specially given in an ordinary statement, this loss is available in the knowledge of the percentage recovery.

Additional to the cost per ton treated, the cost incurred per unit-weight of metal recovered is sometimes given; thus, gold being sold by the ounce, the cost per ounce of gold recovered is given, the cost per pound of copper, the cost per ton of tin, and so on. Such a reckoning of cost declares at once the margin between cost and revenue, that is, the profit; in that, it serves a good purpose. But, being dependent upon the richness or poorness of the ore, it is no direct index to the care and activity exercised; since also the different metals have different unit-weights, such figures of cost afford no comparison of the efficiency obtaining on plants treating different metals; moreover, by dressing it is unusual to produce metal. Accordingly, all things considered, the cost of dressing is best expressed per ton of ore treated.

The results generally obtained are probably best illustrated by giving those of particular performances. In the following statement such particular performances, for their better comparison, are segregated under heads descriptive of the class of plant.

Simple Water-concentrating Plants.—At Joplin, Missouri, treating ore containing 3—4 per cent of zinc irregularly distributed in a calcareous and cherty gangue, a 55—75 per cent recovery is made in a concentrate assaying about 50 per cent of zinc, the cost of dressing being 15*d.*—24*d.* per ton.

Similar work is done in the adjoining lead-district of south-east Missouri, where, from an ore containing about 4 per cent of lead irregularly distributed in a dolomitic gangue, a concentrate assaying 70 per cent is made at a recovery of about 80 per cent, and a cost of about 2*s.* per ton.

On the Alaska goldfield, where the gold is associated with galena and pyrite in small quartz stringers traversing immense bands of slate and gabbro, these bands constituting the ore body, the ore assaying perhaps 1·5 dwt. is crushed and concentrated at a cost of about 1*s*. per ton, with a recovery of about 85 per cent, the tailing from this treatment being discarded.

At Mount Bischoff, Tasmania, a tin ore consisting of soft granite impregnated with about one per cent of cassiterite and some pyrite, is crushed and concentrated for a cost of about 1*s*. 3*d*. per ton, the recovery being about 65 per cent.

The simple washing of haematite iron-ore on the Mesabi Range, Minnesota, costs but 4*d*.—6*d*. per ton.

Complex Water-concentrating Plants.—In the Lake Superior copper-district—where the ore consists of lumps and grains of native copper in an amygdaloidal rock or in the cement of a rhyolitic conglomerate, the copper in the amygdaloid being relatively coarse, that in the conglomerate being finer — dressing of the amygdaloid consists in stamp-crushing, jigging, and tabling, with regrinding of the jig-middlings and sometimes of the tailing. Such treatment recovers about 20 lb. of copper per ton in the form of a metallic concentrate, and leaves about 5 lb. in the tailing, the cost being about 24*d*. per ton. Dressing the conglomerate at the Calumet and Hecla, the tailings from the jigs and primary tables are reground and tabled, after which the sand is leached, while the slime, original as well as reground, is floated; the total recovery of this complete treatment is about 92 per cent from ore containing about 33 lb. of copper, the total cost being about 3*s*. per ton. Leaching costs about 20*d*. per ton leached, and makes a 75 per cent recovery; flotation costs about the same for a recovery of about 65 per cent.

At the Bunker Hill and Sullivan mine, Idaho, treating ore consisting of galena in a gangue of siderite and quartz, by water without the aid of flotation, the results in 1918 were as follows:

	Lead.	Silver.
Ore (feed)	10·2 per cent.	3·8 oz.
Concentrate	66·6 ,,	23·0 ,,
Recovery	84·4 ,,	79·0 per cent.
Milling Cost	2*s*. 6*d*. per ton.	

In Cornwall—where the tin ore consists of cassiterite finely distributed, partly in a siliceous veinstone and partly in granite and altered slate, the cassiterite being sometimes accompanied by wolframite, pyrite, mispickel, etc.—the recovery from ore assaying about one per cent of metallic tin, is about 66 per cent, that of the wolfram when present being about 50—60 per cent, and that of the arsenic in the mispickel perhaps about 50 per cent. Dressing of the complex ore costs about 12*s*. per ton, and that of the clean tin ore about 10*s*. per ton.

Water-and-flotation Plants.—At the Butte-and-Superior zinc mine, Montana, treating the coarse material by water and the fine by flotation, the results obtained during 1919 were as follows:

Assay of ore	15·9 per cent zinc.	6·33 oz. silver.
Assay of concentrate	54·0 ,, ,,	22·00 ,, ,,
Recovery of zinc	97 per cent.	
Milling cost	11*s.* per ton.	

At the Inspiration mine, Arizona, treating the disseminated ore characteristic of the south-west copper-region of the United States, part of the ore being oxidized, the results obtained in 1917 from primary flotation and secondary water-concentration were as follows:

Assay value of ore	1·39 per cent copper.
,, ,, flotation concentrate	35·57 ,, ,,
,, ,, water concentrate	13·93 ,, ,,
Recovery calculated from assays only	75·34 ,,
,, from assays and weights of feed and concentrate	75·78 ,,
,, from assays and weights of concentrate and tailing	75·46 ,,
,, from assays and weights of feed and tailing	75·36 ,,

At the Miami, another of the disseminated or 'porphyry' copper mines, the introduction of flotation in 1914 to supplement water concentration, increased the capacity of the water plant and gave a cleaner concentrate, thus:

	January 1914.	January 1917.
Tons treated	102,497	171,882
Assay of ore	2·31 per cent Cu.	2·15 per cent Cu.
Assay of concentrate	38·97 ,, Cu.	43·14 ,, Cu.
Assay of tailing	0·747 ,, Cu.	0·592 ,, Cu.
Recovery	70·05 per cent.	73·48 per cent.

In respect to these disseminated or 'porphyry' copper mines in general, the following results pertaining to the year 1915 are instructive, since at that date flotation had not been adopted at those mines appearing to the left in the statement. The relatively low recoveries throughout are largely due to the presence of oxidized copper. The low assay-value of the concentrate at the Nevada Consolidated is explained by the fact that this company possessed its own smelter.

'PORPHYRY COPPERS' IN 1915

	Utah.	Chino.	Ray.	Nevada.	Miami.	Inspiration.
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
Feed, Assay	1·43	2·15	1·67	1·54	2·17	1·70
Concentrate	19·17	21·55	19·29	7·77	41·91	32·07
Recovery	64·13	66·59	64·11	70·18	75·17	79·95
Milling Cost	17*d.*	27*d.*	26*d.*	..	..	26*d.*

At the Anaconda, Butte, Montana—where the ore is lode-quartz with some granitic wall-rock, the whole containing about 3 per cent of copper all in the form of sulphide—a concentrate representing about 25 per cent of the weight of the ore is recovered by water from the material larger than 2 mm., and another 11·50 per cent by flotation from the reground material and the original slime. The final tailing assays 0·15 per cent of copper, the recovery being 96 per cent.

At the Braden, Chili, in October 1915, the crude ore treated was 120,000 tons assaying 2·15 per cent of copper, and the concentrate obtained, 10,700 tons assaying 19·2 per cent, this concentrate representing a recovery of 80·8 per cent; the tailing assayed 0·42 per cent. The ore at this mine consists of copper sulphides in veins and stringers traversing shattered andesite. The relatively low recovery is due to the presence of oxidized copper-minerals; in this and in general, the ore resembles that of the disseminated deposits of the United States.

At the Sulphide Corporation, Broken Hill, New South Wales, the concentrates in 1910–1911 assayed much as follows:

Lead concentrate . . .	61·3 per cent. Pb.	8·8 per cent Zn.	28·1 oz. Ag.
Zinc	10·9 ,, Pb.	42·7 ,, Zn.	14·7 oz. Ag.

The recoveries represented, respectively, by the total weights of the metals in these concentrates, and by the weights actually paid for, were as follows:

Apparent recoveries .	93·2 per cent Pb.	87·8 per cent Zn.	90·2 per cent Ag.
Paid-for recoveries .	74·5 ,, Pb.	63·6 ,, Zn.	60·9 ,, Ag.

At that time the lead and silver in the lead concentrate were paid for at market rates, but nothing was paid for the zinc. In the zinc concentrate, after deduction of eight units, the zinc was paid for at market rate, the lead at 1*s*. per unit after deduction of eight units, and the silver at 1*s*. per ounce after deduction of 5 ounces, the last two rates being considerably below the respective market rates.

The concentrates now obtained at Broken Hill are cleaner, and the losses on this account, though still appreciable, are lower. Present costs on that field may be illustrated by the following figures pertaining to work at the British Broken Hill:

	Mining.	Lead Mill.	Zinc Mill.
June 1918	20/8	7/–	5/2
December 1918	22/4	7/8	5/6

In the lead mill the whole ore is treated, chiefly by water concentration; in the zinc mill the zinc tailing is treated, chiefly by flotation.

At the Falcon mine, Rhodesia, the ore, consisting of chalcopyrite with pyrite and pyrrhotite in a siliceous gangue, and containing 2·5 per cent of copper and 6 dwt. of gold, is stamped, tabled, reground, and floated, the concentrates, after sintering, being smelted in blast furnaces. Water concentration, including blankets to catch fine gold, recovers 60 per cent of the gold and 20 per cent of the copper; flotation recovers 20 per cent of the gold and 70 per cent of the copper, the total recoveries being 80 per cent and 90 per cent respectively. More precisely, the figures for the year ending June 1918 were:

Assay-value of the ore . . .	2·2 per cent Cu.	5·1 dwt. Au.
,, water concentrate . .	5·2 ,, Cu.	20·4 dwt. Au.
,, flotation concentrate . .	9·8 ,, Cu.	5·8 dwt. Au.
Recoveries	87 per cent	81·9 per cent.
Smelting recoveries	90 ,,	96 ,,

The oxidized ore at this mine contains too little copper to be worth recovering, but too much to permit cyanidation; the gold accordingly is recovered by concentration on tables and blankets.

At Mount Morgan the ore too siliceous for direct smelting is first concentrated. Typical ore of this character consists of about 6 per cent chalcopyrite, 16 per cent pyrite, and 78 per cent of quartz, calcite, etc., and contains about 5 dwt. of gold per ton. The treatment followed in 1916[1] was as follows: dry-crushing in ball-mills to 50 mesh; table concentration; tailing reground to 80—120 mesh and floated. The following later results, pertaining to the year ending June 1918, appear to indicate a different treatment:

	Tons.	Assay.		Recovery.	
		Cu.	Au.	Cu.	Au.
		Per cent.	Dwt.	Per cent.	Per cent.
Ore treated	123,854	2·29	5·10	..	..
Jig concentrate . . .	20,508	2·60	4·10	18·83	13·31
Table concentrate . .	25,071	3·70	8·51	32·72	33·79
Flotation concentrate .	6,295	18·66	29·18	41·46	29·09
	Total recoveries			93·01	76·19

Flotation Plants.—Beneficiation of an ore directly and exclusively by flotation was first accomplished at the Engels copper mine, California, where, after water concentration had been unsuccessful, flotation was applied in the year 1912; the ore at this mine consists of copper sulphides associated with magnetite in an eruptive rock. The results now being obtained are illustrated by those pertaining to February 1921, when 820 tons of ore assaying 2·7 per cent of copper were treated per day, producing a concentrate assaying 30 per cent and a tailing assaying 0·5 per cent, the recovery being about 83 per cent. The cost of this flotation was 3*s*. 6*d*. per ton, and the power consumption 23 k.w. hour per ton.

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[1] Shellshear, *Aust. I.M.E.*, June 1916; *Abst. E. and M.J.*, Oct. 21, 1916.

At the Swansea mill, Arizona, where the ore consists of chalcopyrite in a gangue consisting chiefly of haematite, breakers and ball-mills prepare for direct and exclusive flotation. The ore assays 2·3 per cent of copper, the concentrate 23 per cent, and the tailing 0·05 per cent, so that the recovery is 98 per cent. The mill has a capacity of 300 tons per day, and the milling cost is about 4*s*. per ton.

Cyaniding Plants.—At the Tonopah-Belmont, mining siliceous-silver lodes in andesite, all the ore is reduced to slime and cyanided. From ore assaying 5 dwt. of gold and 22 oz. of silver, 95 per cent of the former and 93 per cent of the latter are recovered at a cost of 9*s*. per ton. The power installed was at the rate of 1·66 h.p. per ton of daily capacity.

At Santa Gertrudis, Mexico, treating a similar ore, the combined recovery of gold and silver in the year 1917–1918 was about 88 per cent. At Dos Estrellas the milling costs were about 6*s*. per ton.

Milling and cyaniding on the Witwatersrand costs 2*s*. 6*d*.—4*s*. per ton, somewhat less than half the ore being reduced to slime, the remainder being satisfactorily cyanided in the condition of sand. On this field the recovery of the gold is now about 95 per cent, amalgamation being responsible for 50—60 per cent.

Concentrating-and-cyaniding Plants.—Water concentration is applied ahead of flotation, where a substantial proportion of the valuable content may be recovered in concentrate acceptable to the smelter, or, exceptionally, where minerals harmful to cyanidation must be removed.

At the Nipissing silver mine, Cobalt, Ontario, by concentration and cyanidation a recovery of 93 per cent was made where water concentration alone would have recovered only about 80 per cent. The cost of this complete recovery was as follows :

	s.	*d*	
Labour	3	0	per ton.
Supplies	6	6	,,
Power	2	6	,,
General	0	6	,,
Total cost . . .	12	6	per ton.

At the Hollinger gold mine, Porcupine, Ontario, the ore treated in 1914 assayed 13·5 dwt. per ton and the tailing 0·60 dwt., so that the recovery was about 95 per cent. At that time the mill had a capacity of about 600 tons per day, and the cost was about 5*s*. 3*d*. per ton.

Magnetic-concentration Plants.—At Mineville, New York, where the dry magnetic-concentration of magnetite ore assaying 30—35 per cent of iron is undertaken, the concentrate assays 64 per cent and the tailing 6 per cent, the recovery is close upon 90 per cent, and the milling cost about 3*s*. per ton. The cost of the magnetic treatment of zinc ore at Franklin Furnace, New Jersey, where though dry concentration is practised the ore has all to be reduced to 60 mesh, is considerably higher. So also is the cost in wet magnetic plants.

Smelting Plants.—As indicated at the outset, the cost of smelting crude ore is generally about 20*s*. per ton and the recovery 90—95 per cent. Such figures apply to copper ores and lead ores; the crude ores of zinc, tin, etc., are not smelted. Better figures are sometimes obtained where the ore is self-fluxing. Before the War, reverberatory copper-smelting at the Highland Boy, Utah, cost 6*s*. per ton; blast furnace smelting at the Tennessee cost but 4*s*. per ton, and at the Granby Consolidated but 5*s*. per ton.

On the other hand, recoveries by smelting have not always been so good. In the early days of the Copper Queen, Arizona, for instance, the slags from ore assaying 10 per cent of copper, reported as assaying one per cent, were afterwards found to contain 2½ per cent, and the losses by smelting were accordingly of the order of 15—25 per cent.

The recovery in smelting copper concentrate is now generally about 95 per cent; that of lead concentrate probably about the same; and that of tin concentrate probably about 98 per cent. Much depends upon the cleanliness and character of the particular concentrate.

The recovery in smelting zinc concentrate is, however, lower, that from a concentrate assaying 60 per cent of zinc being generally about 80 per cent.

The investment cost of smelting reckoned in terms of the ton smelted, is considerably higher than that of dressing, the total cost of smelting plants being generally of the order of £250 per ton of daily capacity.

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